STRATA CONTROL TECHNOLOGY FOR MASS EXPLOITATION OF UNDERGROUND COAL DEPOSITS: A CASE STUDY OF CONTINUOUS MINER

A THESIS SUBMITTED IN PARTIAL FULFILLMENT
OF THE REQUIREMENTS FOR THE DEGREE OF

MASTER OF TECHNOLOGY (RESEARCH)
IN
MINING ENGINEERING

BY

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ROLL No. 608MN802

Department of Mining Engineering
National Institute of Technology
Rourkela
2013

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Under the Guidance of
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2013

DEDICATED

TO

MY MOTHER

National Institute of Technology

Rourkela

CERTIFICATE

This is to certify that the thesis entitled, "**STRATA CONTROL TECHNOLOGY FOR MASS EXPLOITATION OF UNDERGROUND COAL DEPOSITS: A CASE STUDY OF CONTINUOUS MINER**" submitted by Mr. Sanjay Kumar Singh (Roll No. 608MN802) in partial fulfillment of the requirement for the award of Master of Technology (Research) Degree in Mining Engineering at the National Institute of Technology, Rourkela (Deemed University) is an authentic work carried out by him under my supervision and guidance.

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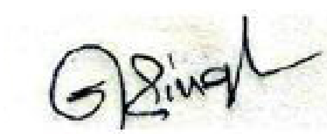
DECLARATION

I hereby declare that research thesis entitle **"STRATA CONTROL TECHNOLOGY FOR MASS EXPLOITATION OF UNDERGROUND COAL DEPOSITS: A CASE STUDY OF CONTINUOUS MINER"**which is being submitted to the National Institute of Technology, Rourkela for the award of the Degree of **Master of Technology(Research)** in Mining Engineering is a bonafide report of the research work carried out by me. The material contained in this research thesis has not been submitted to any University or Institution for the award of any degree.



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ACKNOWLEDGEMENT

The most pleasant point of presenting a thesis is the opportunity to thank those who have contributed to it. Unfortunately, the list of expressions of thank no matter how extensive is always incomplete and inadequate. Indeed this page of acknowledgment shall never be able to touch the horizon of generosity of those who tendered their help to me. First and foremost, I would like to express my gratitude and indebtedness to Prof. (Dr.) S. Jayanthu for his kindness in allowing me to do work in the present topic and for his inspiring guidance, constructive criticism and valuable suggestion throughout this project work. I am sincerely thankful to him for his able guidance and pain taking effort in improving my understanding of this project.

I would like to thank my research Co-supervisor Mr. Gopal Singh, Director (P&P),(Then Their), South Eastern Coal Fields Limited, Bilaspur for giving me this great opportunity to work on this thesis and for his invaluable contribution, and encouragement without him this thesis could never have been put together. Prof. (Dr.) B.K. Pal and Prof. (Dr.) D. P. Tripathy, Deptt. Of Mining Engineering, N.I.T., Rourkela for their personal support and encouragement. Prof. (Dr.) M.K. Mishra, Deptt. Of Mining Engineering, N.I.T., Rourkela and Mr. S.K.Jena, Sr. Manager(M) at Strata Control Cell, South Eastern Coal Fields Limited, Bilaspur for reviewing this thesis and for their invaluable suggestions and recommendations throughout the thesis. An assemblage of this nature could never have been attempted without reference to and inspiration from the works of others whose details are mentioned in reference section. I acknowledge my indebtedness to all of them.

Most of all to my beloved wife Alpana, I thank you for your patience and encouragement. Without your support, I would have never made it to the end. All my love and wish to my daughter Shachi & son Shivansh. I would also like to express my sincere gratitude and deep respect to my father Shri R. B. Singh & my mother late Lalita Devi.

And my sincere thanks to all my friends who have patiently extended all sorts of help for accomplishing this undertaking.

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ABSTRACT

Field investigations were carried out over a two-year period at NCPH Colliery of S.E.C.L; R-6 Mine site for evaluation of strata behavior during extraction of coal in a 6.5 m thick seam by continuous miner at a depth of 106 m. Numerical and empirical models were also used for modification of existing support system leading to formulation of guidelines for the Strata Management.

For the geomining conditions of R-6 mine, maximum of 16 mm roof convergence was observed during widening of galleries. Conventional support system of cement grouted roof bolts of 1.5 m length, 22 mm diameter at a spacing of 1.5 m between rows and 4 bolts in a row. With this conventional support system, widening of galleries up to 6 m has shown no considerable convergence but greater than 6 m wide galleries has resulted in the formation of undulated roof and floor conditions.

Based on field observations including convergence of development and depillaring galleries and numerical modeling studies, the support system was modified with resin bolting so as to provide safer working conditions. The modified support system has 1.8 m long resin bolt for split galleries, 2.4 m long resin bolts for original gallery and point-anchored rebar at 1.5 m center-to-center spacing for the roof conditions of NCPH mine. Based on Numerical modeling results, the bolting was found efficient at a distance of 0.6 m from the side of the pillar and 1.5 m distance from the adjacent bolt.

Although this work is based on studies carried out for the geomining conditions of the NCPH Colliery; R-6 Mine, it is believed that the findings can be applied to other shallow depth coal mines in similar geological conditions. A significant improvement in safety, productivity, and economy was observed at the NCPH Colliery R-6 Mine by adopting many of the recommendations, and strata management guidelines developed through this work.

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LIST OF ABBREVIATIONS AND SYMBOLS USED

ASL: Applied Support Load

BCCL: Bharat Coking Coal Limited

BIS: Bureau of Indian Standards

B.S.: British Standard

CGSST: Coarse Grained Sand Stone

CIA: Combined Instruments Approach

CIL: Coal India Limited

CM: Continuous Miner

CMP: Continuous Miner Panel

CMRI: Central Mining Research Institute

CMR: Coal Mines Regulations

2D: Two Dimensional

3D: Three Dimensional

DGMS: Directorate General of Mines Safety

ECL: Eastern Coalfields Limited

FGSST: Fine Grained Sand Stone

FLAC: Fast Lagrangian Analysis of Continua

Ft: Feet

GDK: Godavari Khani

GED: Goaf Edge Distance

ISRM: International Society for Rock Mechanics

Lb: Pound

LE: Level East

LVDT: Linear variable differential transformer

MGSST: Medium Grained Sand Stone

MSST: Massive Sand Stone

MT: Million tonnes

MTPA: Million tons per Annum

MPa: Mega Pascal

NGI-Q: NGI Rock Mass Quality Classification

NCPH: North Chirimiri Pondary Hills

ND: North Dip

NIRM: National Institute of Rock Mechanics

OMS: Out Put per Man shift

PoS: Possibility of stability

RCI: Remote Convergence Indicator

RLH: Rock Load Height

RL: Reduce Level

RMR: Rock Mass rating

RMT: Rock Mechanics Technology

RQD: Rock Quality Designation

SCCL: Singerani Collieries Company Limited

SCT: Strata Control Technology

SDL: Side Discharge Loader

SECL: South Eastern Coalfield Limited

SEPT: Standard Encapsulation Pull Test

SLD: Support Load Density

SMP: Strata Management Plan

SSR: systematic support of roof

T: Tones

TMT: Thermo Mechanically Treated

TRA: Technical Risk Assessment

UDEC: Universal Distinct Element Code

UG: Underground

USA: United States of America

VK: Venkateshkhani

WCL: Western Coalfields Limited

C: Roof to floor convergence **C_1:** Daily convergence at a site in a day n **C_2:** Average daily convergence at the site up to the previous day i.e. up to day (n-1) **C'_1:** daily convergence on a day (prior to day n) **C4:** convergence station at 0.5 m Anchor **C4A:** convergence station 2.5m Anchor **D:** Distance in m from the line of extraction **P:** required bolt capacity **U:** unit weight of the rock; **t:** thickness of suspended rock; **n:** number of bolts per row; **W_e:** entry width; R: Row spacing; **SF:** safety factor. **h_t:** Rock load height, m **RMR:** Rock Mass Rating (R) **σ_h:** horizontal in situ stress **σ_v:** vertical in situ stress	**υ:** Poisson's ratio **β:** coefficient of thermal expansion **E:** Young's modulus of the rock, **G:** thermal gradient **H:** depth of cover **K:** In situ stress ratio **η:** Maximum deflection (m) **ρ:** Density (kg/m^3) **g:** Gravity (m/sec^2) **E:** Modulus of Elasticity (N/m^2), **t :**Thickness of layer (m) **L:** Span width (m) **C_m :** Maximum ground movement (mm) **B:** Roadway width (m) **γ :**Rock dry density, (kg/m^3) **V_R:** Critical velocity (mm/d) **B:** Roadway width (m) **Γ:** Rock dry density, (kg/m3) **R:** Rock Mass rating **L:** Span (bord width or intersectional diagonal width) (m) **t_{com} :**Competent layer thickness (m) **t_{lam} :**Laminated lower strata thickness (m)

CHAPTER-1

INTRODUCTION

INTRODUCTION

1.0 General

A developing country like India has ever-growing thrust on faster economic development. As energy is the lifeline of all economy, India is genuinely concerned about its energy security. To meet the projected energy demands, Government has declared that fossil fuels, particularly coal, are going to be the mainstay fuel for power generation. Coal provides the single most vital input for the growth of Indian industry. It is the key Contributor to the Indian energy scenario. Out of the four major Indian fuel resources i.e. oil, natural gas, coal, and uranium, coal has the largest domestic reserve base, and the largest share of India's energy production. The most economical method of coal extraction from coal seams depends on the depth and quality of the seams, and the geology and environmental factors. Coal mining processes are differentiated by whether they operate on the surface or underground. Most of coals extracted from both surface and underground mines and it's depending on the techno-economical feasibility. These evaluation are based on the following: regional geologic conditions; overburden characteristics; coal seam continuity, thickness, structure, quality, and depth; strength of materials above and below the seam for roof and floor conditions; topography (especially altitude and slope); climate; land ownership as it affects the availability of land for mining and access; surface drainage patterns; ground water conditions; availability of labour and materials; coal purchaser requirements in terms of tonnage, quality, and destination; and capital investment requirements.

Surface mining and underground mining are the two basic methods of mining. The choice of mining method depends primarily on depth of burial, density of the overburden and thickness of the coal seam. Seams relatively close to the surface, at depths less than approximately 50 m, are usually surface mined.

Coals that occur at depth of beyond 50 m are usually underground mined, but in some cases surface mining techniques can be used. For example, some of mines, coal that occur at depths in excess of 60 m are mined by the open pit methods, due to thickness of the seam 20–30 m. Coals occurring below 100 m are usually deep mined. India is the world's third largest coal consuming nation after China and the USA. Coal is the dominant energy source in India, accounting for more than half of the country's requirements. 70% of India's coal production is used for power generation, with the remainder being used by heavy industry and public use. Domestic supplies satisfy most of India's coal demand. The Working Group for coal & Lignite formulation of XII plan has assessed a coal demand of 980.5 MT by terminal year of XII plan i.e. 2012 – 17 and is projected to 1280 MT by the end of 2024-25. The annualized growth rate of coal demand is expected to be about 9% over. To meet the ever-growing demand for coal in country and to mass exploitation of underground coal seams. Because opencast reserves in the country have been either largely exhausted, or are on the verge of exhaustion and the future holds more promise for UG mining. It is widely considered that lower seams of coal tend to possess better grades and this is what the country is eyeing. Also, those lower seams can be reached and extracted only with UG mining methods. India's coal ministry wants to better utilize land to meet a growing energy demand. The focus would be on making full utilization of underground resources by stressing on underground mining. As per recommendations of expert committee 'the Powered Support Longwall and Continuous Miner technology is being applied with success in many mines and there is a need to popularize and establish these as predominant underground technology especially for mass production. In order to work out an action plan for the coal sector in line with the recommendations of the expert committee on integrated energy policy. It is high time and also there is an urgent need to introduce mass exploitation technology.

1.1 Mass Exploitation of Coal

Mass exploitation of coal refers as "economically excavation of coal with due method of higher productivity, safety and conservation". Present intermediate mechanization, based coal mining may not be suitable to meet the global competitiveness (arising due to open economical policy) of productivity. There is a need of fully mechanized or even an automated underground method for a safe coal mining, which may also strengthen our industry to meet the global competitiveness of productivity.

Mass Exploitation Technology needs large size machines and more space for its maneuverability for better utilization of the machinery. Up-gradation of technology is continuous process to be competitive. It is accepted that mass exploitation is essential not only to reduce human drudgery in manual mining but also for economical survival.

For keeping this view, present suitable Mass Exploitation of underground Technology available in Indian coal mining industry as follows:-

I. Longwall
II. Continuous Miner
III. Highwall
IV. Blasting gallery method

With reference of above technology adoption, strata control problem is one of the major reasons for facing hurdles in safety as well as productivity in underground coal mining is associated with inequalities related ground/strata movement within the host rock geometry. Generally 33% to 43% time spent out of total underground in mining cycle’s operation for strata control activities, it’s depending on geo-mining conditions of the working area. In turn strata movements dictate stability of workings, which remained as the prime concern in underground coal mining with mass production technology. Strata control technology refers to study of many parameters of rock mechanics to fulfill the objective of safe mining with productive exploitation of underground coal deposits. Strata control implies the control of the strata to facilitate mining operation to be done efficiently and safely. This has become an emerging problem in strata control monitoring in Indian underground coal mining with degree of accuracy and reliability because cost of project for mass production/exploitation of underground coal deposits is very high as compare to other available technology. The technology of underground coal mining has influenced by the strata control practices, space constrain, subsidence/environmental problems, managerial skill and efficiency and size of machinery. This study deals with such system of Strata Control Technology for Mass Exploitation of Coal deposit with Blast Free Continuous Miner operation.

1.2 Strata Control Technology

The term "strata control" principally refers to controlling the strata to maintain stability around the mine openings in underground where operations are or will be taking place. The need for strata control may extend into a goaf area for a short distance, essentially to the goaf edge, however strata control within the goaf is generally of no interest. In order to analyze strata reactions, properties such as strength (tensile and compressive), modulus of elasticity, Poisson's ratio, etc are required, as well as details of the likely stress fields to which they will be subjected.

If these are unknown or cannot be measured, then its value is assumed with excessive conservative designs likely to result. A reasonably detailed knowledge of any geological structures is also required as these can affect both strata properties and stress fields locally. Strata control techniques which are used include:

- Mine design relating to dimensions and shape
- Mine design relating to mining direction
- Sacrificial support external to strata
- Reusable support external to strata
- Strata reinforcement
- Retention of failed strata

Underground coal mining industries throughout the world is breaking down slowly. Although, in some countries underground coal mines are being operated, their numbers are very less and decreasing. Most of the big coal mines in the world are being operated by open cast mining method. But open cast mining has got some limitations. It cannot be operated when depth of the coal seam is very high. Most of the good quality coal deposits in India are at very high depth and are being operated by underground mining method, which is the only economically viable method of extraction. The days will come back when people will have to think again for underground coal. But due to geological disturbance and adverse geo-mining conditions production from underground coalmines in India could not be enhanced; even after mechanization of some underground coal mines. Main problem in the underground coal mines is the ground control and stability problem; which is nothing but instability caused in any rock structure because of movement of rock in the earth crust. It is one of the causes of roof/side fall that obstructs smooth production from underground coal mine.

The progress of the technology in many branches of engineering is quite rapid in recent years. However, in case of underground coal mining, the progress is not as expected. It remained a lot with traditional systems, and only a few attempts were made to adopt/absorb recent trends. Although it could be attributed partly to availability and adoptability of the modern mining machinery, but also mainly due to limitations of available strata control technology, be in underground (suitable designs of workings and support systems). In Indian coalfields, general practice to control strata is to support the excavated area by suitable, efficient and necessary, means. In normal practice excavated galleries are supported by some means without completely studying and analyzing the behavior of rock causing fall of roof and side in underground coal mines in India. Due to roof and side fall, there have been a number of fatal accidents in underground coal mines. Strata control is a major problem, which affects safety and productivity in underground mining. Roof fall is a cause of uncontrolled strata. The primary causal factor for poor roof is presence of week bedding planes. Geological disturbances such as joints, slips and faults, rank second in importance. Shale in the roof of coal seam is responsible for deteriorating roof condition primarily due to weathering of the type of rock in contact with water or humid mine atmosphere. Strata control deals with the adaptation of a system by which we could have a control on the strata movement to a desired level to make our workings safe and extraction of coal possible.

1.3 Continuous Miner in Underground Coal Winning

Mass exploitation technology using Continuous Miner (CM) is one of the suitable alternatives for Indian coal mines in order to efficiently boost the coal production from underground mines. The scenario of a higher production share from surface mines is not going to be sustainable because of reduced near surface coal reserves and other concerning issues attached with surface mining. Considering these restrictions the two state owned coal companies, Coal India Limited (CIL) and Singareni Collieries Company Limited (SCCL), have taken a lead to boost the coal production from underground mines through CM mining technology.

At present five mines under different geo-mining conditions are extracting coals from previously developed square pillars with CM technology and the majority of them experienced unexpected roof fall incidents perhaps due the geo-mining conditions that were not appropriately anticipated and accounted during the planning stage. Four of the mines are using the pocket-and-fender method for coal extraction which is the least favoured method with CM technology due to safety reasons (Mark, *et al*., 2002). Five mines are developing coal blocks using CM technology. Three of the mines introduced CMs with a cutting drum width of 3.3 m and two have cutting drum widths of 2.7 m. This means that for economical reasons two mines shall operate with 5.4 m wide rooms and rest of the mines operate with 6.6m room width OMS from all these mines has shown a threefold to tenfold increase in comparison to the conventional mining practices and there is potential to further increase productivity from these mines should proper geotechnical planning be considered for the final extraction program. There are five mines, namely - GDK11, Tandsi, Kumbharkhani, Rani Atari and Chirimiri, operating with continuous miner technology in India where creations of rooms is being undertaken. Additionally, the Western Coalfields Limited (WCL) will implement continuous miner technology at its more underground (UG) mines apart from the operating two mines of Tandsi and Kumbharkhani in two phases. The new method is more machine-oriented than the conventional mining method involving drill and blast cycles. Two of the operating mines have CMs with cutting drum width as 2.7 m implying that economic reasons dictates room width shall be at least 5.4 m while the other three mines have CM cutting drum width at 3.3 m giving the possibility for 6.6 m wide rooms.

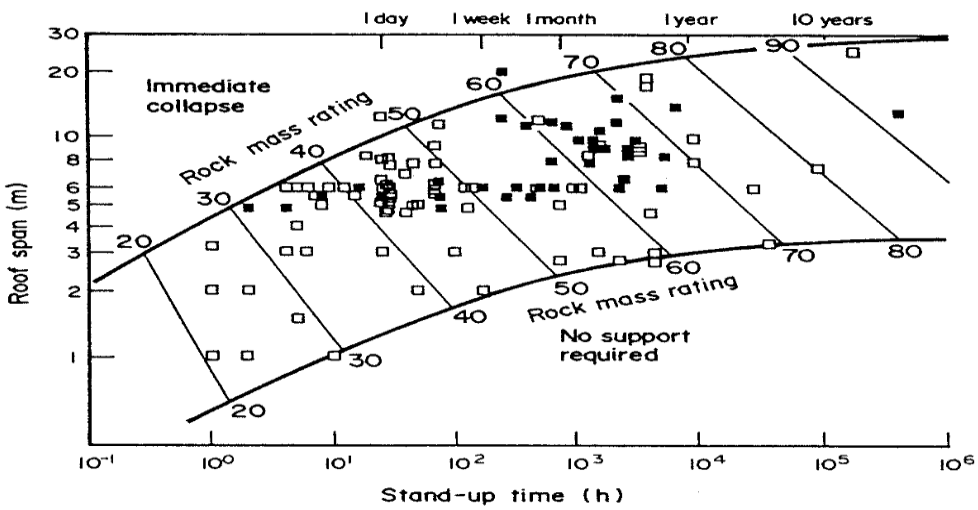


Figure- 1.1: Relationship between the stand-up time and span for various rock mass classes according to the RMR system

Geo-technical conditions dictating the room width can easily be ascertained by the stand-up time concept given by Bienawski (Bieniawski, 1989). Figure-1.1 illustrates the stand-up time concept with Rock Mass Rating (RMR) values plotted on it for some of the operating mines and planned mines. The statutory permitted room width for Rani Atari and Kumbharkhani mine is 5.4 m while Tandsi Mine is forced to work under 4.5 m room width due to poor geo-technical conditions. Chirimiri and GDK11 mine are permitted for 6 m wide room creation. Our earlier study reveals that the decision to introduce CM with 3.3 m wide cutting drum for Tandsi mine was not a proper decision. The mine has a severe issue of ground control related problems caused by high horizontal stresses and a solution to deal with the stress regime should be addressed along with the creation of rooms. A proper study prior to introducing the CM technology would have helped the mine management. It also suggests that the room widths of more than 6 m with a cut-out distance of 12 m can easily be operable parameters for the planned mines except the Nand I Mine. Rani Atari and Kumbharkhani mine has developed more than 20 km of development in the respective mines without an incident related to roof fall and both the mines used the stand-up concept to design the room width. The concept dictates that the maximum room width shall be designed in such a manner that the roof shall not fall within a period of 48 h prior to installation of the rock reinforcement measures. The critical time period of 48 h is kept in case the reinforcement measures could not be applied due to some technical problems in the mine.

1.4 Description of the Study Area

NCPH Colliery, R-6 mine, located in Chirimiri in Korea District (C.G.), is under Chirimiri Area of South Eastern Coalfields Limited (SECL).In this mine, No. 3 Seam (3-3.2 m thick) dipping 1 in 10 is developed by Board and Pillar Method. The galleries are 4.5 m wide and 3 m in height. Pillars are 23.5 m x 23.5 m (corner to corner) and 22 m × 22 m (centre to centre). NCPH colliery is situated in almost central part of Chirimiri coalfield. The colliery is working four coal seams in its two parts, which are separated by a major fault. It is working with continuous miner to depillar S-I panel of No. 3 seam with caving. The depth of cover working seam varying from 60m to 106m. A few cases of roof falls have been reported in the developed workings of No. 3 seam mainly at the junctions. The maximum height of these falls is around 2 m at 3LE/15DN and 5LE/16DN junction. Side spalling is also observed at few locations leading to the widening of existing galleries.

1.5 Objective of Study

The Objective of the study presented in this thesis is to improve the understanding of the fundamental mechanisms of roof behavior and the essential of support design and a safety based design methodology for their amelioration. To meet the main objective of the study, these are the primary objective of this study is to:

- ❖ Study of Strata Behavior with respect to convergence during extraction of coal in a thick seam (6.5 m) by continuous miner with diagonal slicing.

- To verify the suitability of existing support system for ensuring safety based on field observations and numerical models.
- Formulation of guidelines for Strata Management.

1.6 Methodology of the Present Study

The above objectives could only be reached if acted upon with a planned approach. The first step towards a goal always starts with knowing everything about it. Thus we began with the literature review. The books, journals, papers proved a rich source of knowledge in this regard and were thoroughly studied and learned. Discussion with officials encouraged us further in our work. This was followed by extensive field investigation & collection of data from the field site. The geological data collected were location of seam, depth of seam, seam thickness etc and the mining data collected were borehole data, pillar dimensions etc. Failed and stable case histories were also studied.

Almost eighteen months field data collected from the NCPH Colliery, R-6 mine of South Eastern Coalfields Limited (SECL) and carefully analysis. Then we had modified the existing support system with empirical assessment and numerical models based on the safety factor. After that strata behavior observed and evaluated without compromising the safety factor. Finally we conclude the setting of guidelines for support system in continuous miner operation.

1.7 Constraint Associate with the Study

The study in question may suffer because whichever instruments used for strata behavior monitoring they have some limitations. In this project, we have field investigation of strata behavior with respect to convergence of depillaring working with existing support system at NCPH Colliery, R-6 mine of South Eastern Coalfields Limited (SECL). These are applied for analysis of existing support system and verification of the support system based on field observations each one have separate boundaries in practical way of implementation and setting of guidelines. These are the following limitations associated with present study are:

- Uncertainties in strata behavior due to variability's of material properties.
- Inconsistencies associated with instrumentation plan during field investigation regarding magnitude and direction of station.

- Uncertainties associated with empirical assessment and numerical models due available soft ware system.
- Inaccuracies that may be arise from the modified support system.

1.8 Outline of the thesis

Following this introduction, detailed literature review on the subject is presented in chapter 2. Current knowledge in the fundamental of strata control techniques and support system in Bord and Pillar mining for Mass Exploitation of Underground Coal Deposits with continuous miner is summarized.

In chapter 3, a detailed underground field investigation programme was carried out in NCPH Colliery; R-6 mine situated Chirimiri Area of South Eastern Coalfields Limited (SECL) in significantly different sequence of working environments. The observations of strata behavior on the roof and support performance was also investigated as part of this study. The results from this monitoring programme presented in Chapter 4 with modification of support system and their observation in effect of strata mechanisms.

As the Chapter 4 indicated stability analysis with strength and deformability of rock masses in specified area in underground coal mines with variable nature of the roof behavior, geotechnical classification techniques were evaluated to determine their effectiveness in predicting the variation and uncertainties in the modified support systems and compare with numerical models. Based on the knowledge gained throughout this study a new simulation model of modified support system has been developed in this chapter also. Analysis of this field investigation and support system in study area are given in Chapter 5.The conclusions and recommendations are presented in Chapter 6 with suggestive guidelines for support system in continuous miner operation. References of this study and thesis writing are given in Chapter 7.

CHAPTER-2

LITERATURE REVIEW

LITERATURE REVIEW

2.0 General

Strata control is the science (some would suggest art) of utilizing various techniques to prevent or control failure of the strata around mine openings at least for the period where access is required. For different locations in the mine this period may be for the life of the mine (which can be considered as permanent), such as the main mine accesses from the surface, or for a matter of less than an hour, such as a lift off a coal pillar with a continuous miner. Strata refers to rock in all the possible forms that it may take from a high strength material to an extremely weathered, very low strength, essentially soil like material. Strata control refers to the methods applied to manage the risks associated with various forms of strata instability in underground coal mines.

The aim of this aspect of strata control is to make the strata self supporting as far as possible, or if not, to minimize the extra support work required. With regard to opening size, this involves designing minimum practical widths for whatever operations are carried out and could involve modifying the design of equipment to fit into smaller openings. It is perhaps more common to install extra support to stabilize an opening that is suited to available equipment rather than design and manufacture equipment to suit the opening. It is likely that the economics of the latter alternative are seldom examined closely.

Mining sequences can be designed to allow intersections to be mined across existing roadways and minimize breakaways which are always bigger excavations. Equipment still needs to be able to turn the corners however. Opening size is always going to be a compromise between a desire to minimise excavation and maximize stability versus minimizing ventilation resistance and maximizing the available work space. The height of excavations also needs to be considered – is it better initially to mine less than the final working height in a thick seam for the benefit of more stable ribs? In the event of rib failure openings effectively become wider to the depth that the failure extends into the rib. With regard to pillar design, the aim is usually to design pillars large enough to remain stable under increased vertical load caused by redistribution of the load previously carried by the extracted coal. Note that there may be several stages to this load redistribution as first and second workings are undertaken.

There are some cases where pillars are actually designed to yield (i.e. at least partially fail) in order to relieve stress on adjacent roadways. During second workings with continuous miners, remnant pillars or stooks may be designed to remain stable for only a very short time and then be allowed to fail in the longer term (in fact this may be desirable to improve caving).With regard to pillar stability, it is not only the plan area which is of importance but also the height to width ratio – a tall, thin pillar is more likely to fail than a short, fat one. The length and/or width required for a stable pillar is therefore going to increase as the working height increases.

The shape of an opening also affects its stability. A circular opening is the most naturally stable shape and has been used at mines, notably for shafts and drifts. While a circular profile may be more stable a flat floor is required for most purposes – generally there is little point in removing strata in the lower portion only to re-fill it again afterwards. An arch shape provides the benefit of a circular profile in the upper section while retaining a flat floor. The drawbacks of an arch section are:

- Because the width of an arch narrows towards the top, an arched roadway may need to be mined wider and/or higher than a rectangular roadway to obtain the dimensions required for given equipment to pass.
- Typical continuous miners have wide cutting heads to maximize production (coal being soft enough to not require excessive cutting power). Mining an arch section, particularly in stone requires a narrower head machine to excavate the profile and meet potential power limitations.
- In laminated or banded strata, the curved portions of roof often fall away in part so the shape tends to a rectangle of its own accord.
- An arched profile would be incompatible with the current design of longwall gate-end supports.

For these reasons, nearly all mines cut rectangular profile openings, apart from in shafts and drifts. Strata reinforcement is used in almost every mine today to some extent, most commonly in the form of roof bolts. The earliest roof bolts were steel rods with a split end with a steel wedge inserted. The rod was installed into a hole drilled in the roof and hammered in so that the wedge forced the steel to grip the sides of the hole. A nut at the outer end was tightened against a washer and steel plate installed against the roof to apply some tension and, with similar rods being inserted across and along a roadway, the result was that strata beds in the immediate roof were clamped together to form a stronger beam. The bolting pattern density was increased until the roof then became self supporting. Such bolts were often installed through timber bars to spread the support over more area and to aid in retaining broken material. At times steel cross members were used instead of timber, the bolts being installed through brackets or "saddles" to hold them in place.

Over time the split and wedge bolts were replaced by improved methods of anchoring, eventually using fast-setting, two part resin cartridges, the resin being mixed by the rotating bolt as it was inserted in the hole. These bolts were initially anchored at the end of the bolt (referred to as **"point anchor bolts"**), and although tension was applied during the installation process, a lot of movement occurred before the bolts became really effective. It was found that better results were obtained if sufficient resin was used to completely fill the hole drilled in the roof and anchor the bolt over its entire length (referred to as **"full column anchors"**) and these are now universally used for primary roof bolting. The full column anchor also has the advantage of protecting the bolt from corrosion in corrosive conditions.

Roof bolts were often used in conjunction with steel straps in place of timber bars, the straps being stronger and better able to mould into uneven roof, but still were mainly of use in retaining broken roof. Most mines now use mesh sheets which completely cover the roof, held in place by the bolts. They have minimal support function but prevent injuries from falling material.

Other developments which have followed on from normal roof bolts mostly relate to the length of reinforcement placed in the roof. The maximum length of a normal bolt is the working height less the height needed for the roof bolting machine. Initially if longer bolts were required they were made in sections which could be screwed together. Later developments saw flexible bolts (essentially lengths of wire rope), able to bend enough for installation but stiff enough to push up the hole. Various designs of such bolts are now available. They are mostly installed vertically or slightly angled over the rib. They are too long for resin to be placed in cartridges and the normal method of anchoring involves pumping a grout up a tube inserted with the bolt. Sometimes a resin point anchor is installed initially, allowing the bolt to be tensioned before being fully grouted.

A slightly different application for flexible bolts is a **"truss"**. These consist of 2 flexible bolts installed at an angle over opposite ribs of a roadway with long "tails" left in the roadway. The tails of each pair are joined together at roof level and tension applied, so that there is a degree of horizontal compression applied to the roof strata. The aim is to pre-stress the roof to assist in preventing failure, although in many cases trusses have been installed in already failed roof to act like a basket to retain the broken material in place (still a valid strata control function).

The above comments refer to roof reinforcement, but many mines use bolts in the ribs to prevent rib failure, usually in conjunction with mesh. Rib bolts do not need the strength required for roof support, their function being often to retain broken material in place more than to prevent rib failure. Some rib bolts have to be installed in ribs which later have to be mined, with a consequent need to remove the bolts before the coal is put into the coal haulage system, which can be difficult. Various "cuttable" bolts have been developed to allow mining to continue regardless of the presence of such bolts.

On some occasions where floor heave is a problem, bolts are also installed in the floor to help to control movement. Such bolts need to be cut-off or installed completely below floor level to avoid tyre damage if vehicles are required to use the roadway. They can also be a trip hazard if proper precautions are not taken. The reinforcement of strata by injecting various types of resin or cement material into it is now routinely adopted in adverse ground conditions. It is usually quite a slow process and some resins can be a health hazard requiring personnel access to be restricted during pumping. Therefore their use cannot be incorporated into the normal mine development process. The main use of this type of reinforcement is to pre-grout strata where stability problems are predicted e.g. where geological structures are expected, or to grout around areas where a major failure has already occurred to assist in recovery of control. The latter method is quite common where the roof has been lost on a longwall face.

The reinforcement process involves drilling into the strata then pumping the resin or cement material into the hole under pressure, forcing the material into any spaces in the strata and gluing it together. Some of these materials form high expansion foam which also creates its own pressure thereby improving penetration of the strata. At times these materials are used to fill a cavity where a major failure has occurred. In such cases they are not being used so much to reinforce the strata as to replace the strata so that operations can recommence. Care is need in the use of these materials as:

- The pressure applied during injection of grouts can itself cause strata failure.
- Resins generally are a two part mix and the chemical reaction is exothermic. Fires have occurred during placement. There are usually restrictions on the amount of resin which

can be pumped into any one hole and they are not suitable for placing in bulk in cavities.

- Some resin materials are carcinogenic before the reaction has taken place so access to work sites and on the downstream ventilation side is restricted.

2.1 Strata Mechanism

Underground excavations in rock cause redistribution of stress around the opening. Depending on the strength and deformation behavior, the rock adjusts itself by moving into the opening. Sedimentary rocks have low tensile strength normal to the bedding plane, and low shear strength along the bedding planes. Adverse geological conditions in any area may further reduce the overall strength of rock mass. The deformation of roof has added advantages having gravitational forces, hence are liable to failure. Timely and proper support is necessary to prevent collapse of roof.

In layered strata like coal measure rocks, bed separation and subsequent roof sag take place in the immediate roof. Simultaneously, the load originally carried by the coal is transferred on to both the sides of the solid pillar. These “abutment stresses” are much higher than the average pressure on the surrounding area. Figure-2.1 showing details for typical underground excavation (A) of coal seam V.

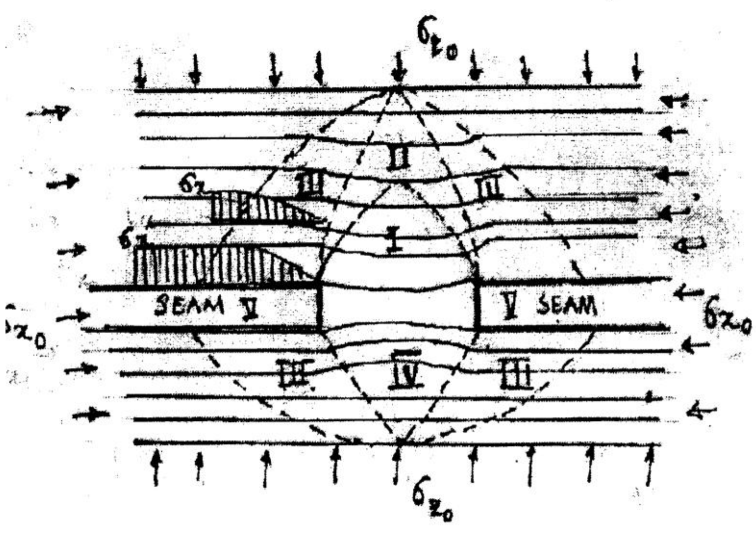


Figure-2.1: Bed separation and distribution of stresses

Zones-I and II are distressed areas. The bed separation in zone one-I gradually reduces towards the top mainly due to clamping action of abutment pressure and frictional resistance between the layers. The arches above the opening depend on the component layers, uniformity in thickness, and the magnitude of horizontal pressure. Depending on the conditions or rock and stress fields, floor heaving and side spalling may also occur.

2.2 Factor Influencing Strata Mechanism

These are the following factors influencing roof stabilities in any underground excavation are as:

- **Span**

  One of the major influencing the stability of roof and support requirement is the width of the roadway. This factor becomes increasingly important with increasing RMR.

- **Profile**

  A curved profile as compared to a rectangular section almost invariably improves the inherent stability of the roof by substantially reducing stress concentration. Theoretical concentration indicates that the tangential stress at the corners of a rectangular opening approaches infinity.

- **Virgins stress**

  To ascertain the total stress condition at any point it is necessary to measure stress value in three conditions. The vertical stress field value is generally given by:

  Vertical stress (MPa) = Depth (m) x 0.024

  However, in-situ measurements are required for assessing the state of stress.

  Lateral stress, if excessive, may cause shear failure of roadway roof. In high lateral stress field, there is a definite best and worst drivages direction. Main roadway directions are to be established accordingly. The preferred line of drivages coincides with the principle horizontal stress. It has been shown that a corridor of approximately 300 either side of the preferred direction exists in which to drive good roadways. It was generally assumed that the lateral stress was 1/3 of the vertical stress. However, in Australia, in particular, it has been seen that the lateral stress is principle stress and is in excess of the vertical stress in the mining areas. In UK also a high lateral stress has been observed in Selby area. In India not much study has been carried out to determine the lateral stresses.

- **Induced stress**

  These are caused by mining activity in another seam or in proximity in the same seam. Induced stress as and when they appear call for secondary support, which should ideally be erected to prevent damages due to the stresses induced.

- **Physico-mechanical properties**

  For, all practical purpose only the compressive strength of roof rock is taken into consideration, except when mathematical modeling is attempted using various other strength parameters, such as tri-axial strength, shear strength etc. Slaking/swelling is an important property. Clay particularly on exposure to moisture display instability and poor anchorage to roof bolt. Joints reduce rock strength, particularly, in tension and shear. The worst drivages direction will be parallel to the major joint plane.

  Depending on throw, clay filling and joint swarms, faults will affect the roof variably and call for substantial up rating of support system. In addition to above there are other structural features, like false-bedding, slickenside, streaks of clay or coal etc.

2.3 Effect of Mining Parameters on Strata Control

In order to design satisfactory strata control measures it is essential first to have a clear about the mechanics of the movement of the ground as a result of mining operation. These are the following mining parameters which affect the strata control conditions are:

- Depth of the seam
- Layout of the face
- Method of treatment of the goaf
- Working thickness of coal seam

Effective strata control has a function of three main components:

- Strata characteristics
- Mine planning and design, and
- Strata control measures.

Strata control is applied at all stages of a mine, however, only where it is suited to the particular characteristics of the mine’s strata, design and layout of the mine can the risks to health and safety be minimised.

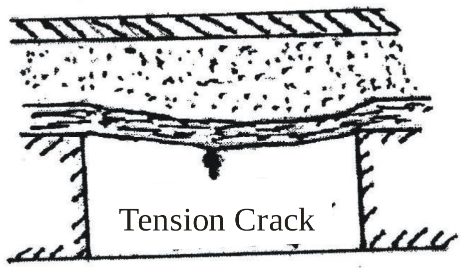
2.4 Reasons of Strata Failure

An understanding of the failure mechanism is necessary to design the support system, which basically attempts to prevent such failure. Usually, failures in mines are brought about by a combination of causative factors. The various modes of failure may be classified as below:

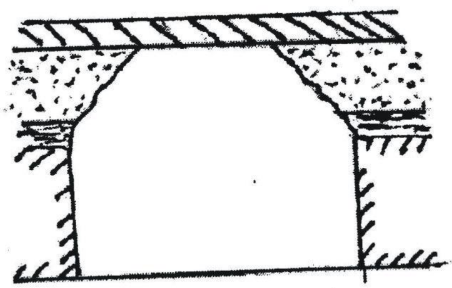
2.4.1 Tensile failure

Tensile stress in strata is generated by the gravity loading of the sagging strata. Cracks form along the edge and the centre of the roadway, when the failure planes join up, the strata cave. These failures occur under low horizontal stress conditions. Typical cases of span failures are shown in Figures. Obvious remedy is to prevent roof sag by reducing span and/or by roof bolting to increase the tensile strength of the roof beam or to provide suspended support. Repeated span failure may end up in an arch failure.

Another form of failure known as "skin failure" can be attributed to tensile failure, but other inherent weakness in the rock mass like friability, cross bedding, slickenside etc.



(a) Span Sag



(b) Span failure

Figure-2.2: Span failure

Contribute more towards such failure. In this type of failure thin layer of immediate roof caves in small segments. Such layers may be dressed down or coal may be left in the roof. Alternatively, bars and/or wire mesh may be used between supports.

2.4.2 Shear failure

These failures are manifestations of lateral stress. Mid-span failures occur under relatively uniform stress field or where beam failure has already weakened the material.

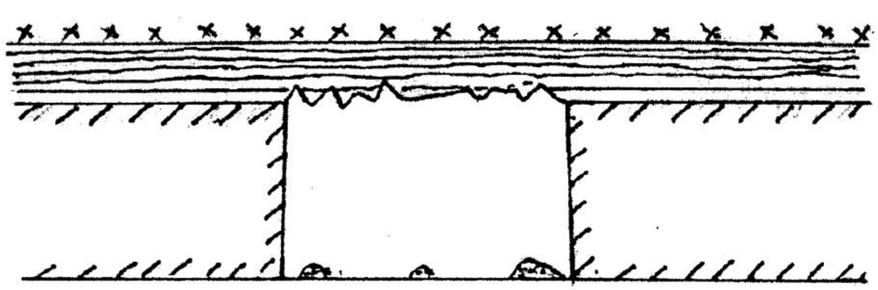


Figure-2.3: Skin failure or flaking or unraveling

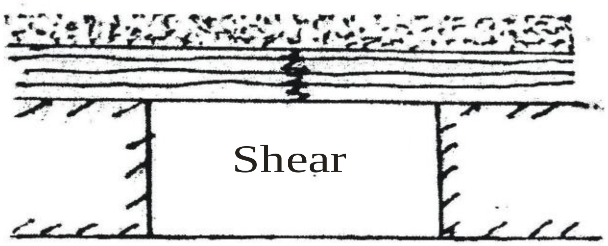


Figure-2.4: Mid-span shearing

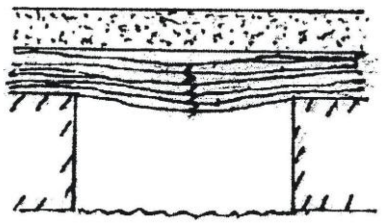


Figure-2.5: Combination mid-sap and beam failure

Shearing may occur along the pillar side when the lateral stress is high. This is the first stage of the failure mechanism and is known as guttering. The shear planes usually extend over the roadway.

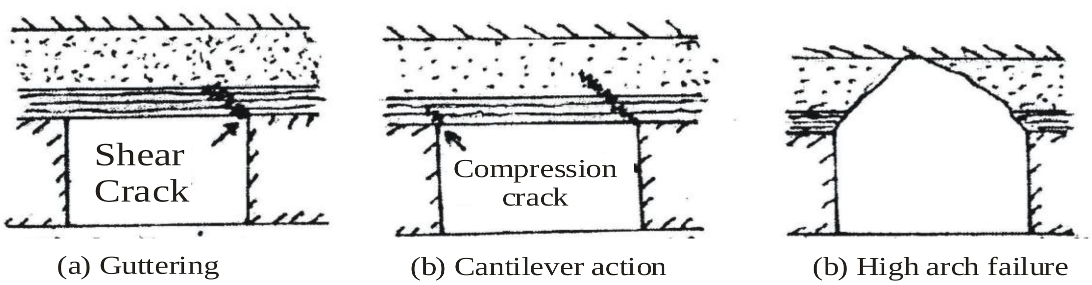


Figure- 2.6: Skin failure or flaking or unraveling

The fractures extend higher into the roof by cantilever action and roof leans towards the shear side. Compression cracks may appear along the other side. This ends up in a high arch failure due to extension of failure planes to higher strata. The progress of shear failure can be arrested at any stage by taking appropriate action. It may stabilize on its own. The remedial measures include aligning of the main drivages according to the orientation of the lateral stress, reducing span and systematic quick erection of support.

2.4.3 Structural failure

These failures are caused by structural defects in the roof rock. These defects bring in discontinuities in the rock mass and it reduces the strength. The most common structural defects are joints, faults, dykes, slickenside false bedding, etc.

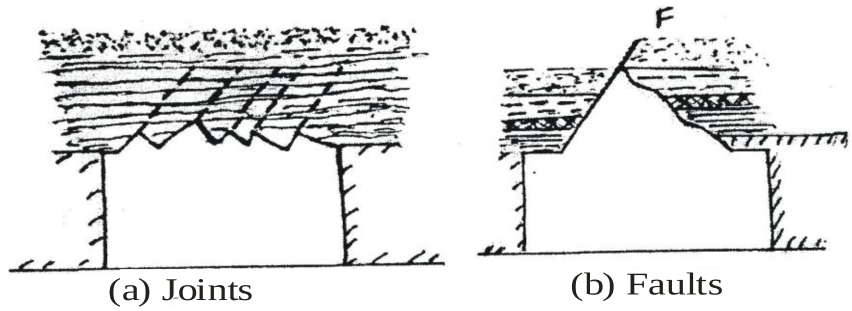


Figure-2.7: Structural failure

2.4.4 Arching action

Arches action is the natural process by which a fractured material acquires a certain amount of ability to support itself partially through the resolution of the vertical component of its weight into diagonal thrust. If support is installed before the initiation of roof separation, it strengthens the ground structurally, and enables it to support itself. The strength of such support is only a fraction of that which would be needed to support the full weight (dead weight) of the roof strata overlying the opening. The supporting force need only be sufficient to prevent failure (by shearing) of the strata under compressive stresses.

In case of an opening overlain by fractured strata, the fractured blocks will be prevented from falling because they are not allowed to rotate about their edges. The restraining forces preventing rotation are simply the general reactions. Friction forces at the end of the blocks resist shear forces and prevent the blocks from moving vertically downwards.

Even if the roof is cut numerous fractures, the result will be the result provided no lateral movement is allowed. This is the reason why opening in moderately fractured rock will stand without any support, and those with badly fractured rock will stand with a minimum amount of support. Such decoupled rocks are supported entirely by compressive and shear resistance, and strength of the linear arch does not depend at all on the tensile or flexural strength of the rock, but depends on the compressive strength of the rock (which is normally at least 4 to 5 times as great as flexural strength in unfractured rock, and infinitely greater in fractured rock). The presence of large lateral stresses tends to stabilize a linear arch.

After understanding the failure mechanism of strata, we can establish numerous techniques and can design specific supports to counteract the predominant failure mechanisms. Keeping in view, as mine manager should be determine and should deploy the appropriate technique to work safely in the mine.

2.5 Special Features of Thick Seam Mining

Board and Pillar mining with development in different horizons of a thick seam has been a popular method of extraction leading to locking of 1,835 million tons of coal in different major coalfields of India. About 70% of these reserves are to be extracted by underground mining. A major portion of these reserves are is amenable for caving without any surfaces structures.

Coal seams is the range of 4.8m to 9m thickness are considered critical due to non applicability of multi section mining and limitations of conventional support systems for single lift extraction as per Indian mining law (Rakesh and Prasad, 1995). As a result, limited extraction height leaving coal in the roof or floor of the seam, not only lead to spontaneous heating and premature sealing of the workings, but also caused adverse strata control problems as a special features of thick seam mining (CMRI, 1997)

Extraction of tick coal seams, in general and seams developed on pillars in particular, has posed serious challenge to the mining engineers in view of strata control problems in openings higher than 4.8 m. As a result, the final extraction has been permitted upto 4.8m height irrespective of excess thickness of the coal seam. Therefore, overall recovery was 30-40 % only by the conventional bord and pillar methods, and decreases drastically with increase in thickness of the seam. Experimental trial in 6.5 m thick seam was conducted to extract full seam height for the first time by S-1 panel at New Chirimiri Pondry Hills (NCPH) colliery, R-6 Mine of South Eastern Coalfields Limited.

2.5.1 Methods of Extraction of Pillars.

Many methods of coal mining were developed in different countries of the world, but options for extraction of developed pillars in thick seams are few. Singh & Dhar (1992) presented and discussed different methods of pillar extraction in thick seams with special reference to the experimental trials in Indian coalfields. Variants of pillars mining were also discussed in different symposia on thick seam mining (Singh, 1998). In view of the unlimited production demand, the methods are influenced mainly by the seam thickness, depth gradient of the seam, quality of the coal and surrounding rockmass.

Seams exceeding 9 m thickness are invariably developed to pillars in multisections and extracted in conjunction with stowing or caving. Pillar extraction was in practice since long to 5 m height and seams upto 8.5 m were also worked in India using timber supports (Singh, 1962). Different methods of extraction of pillars including recent experimental trials are listed in Table 2.1

Table- 2.1: Methods of pillar extraction by caving in thick coal seams (after Singh, 1992)

Seam Thickness (m)	**Method**	**Recovery (%)**	**Remarks**
4.8-6.0	Board and Pillar caving in Single lift.	70-55	Sometimes coal in floor and roof is left.
6.0-7.5	Splitting and stowing in bottom section and caving in the top section.	80-70	The most easy and prominent method.
7.5-10.0	Caving in two lifts.	55-50	A 3m parting in the middle is left.
9-11	Hydraulic mining	70-55	Failure in Indian conditions.
10-12	Blasting gallery	85-75	Successful in fairly good roof conditions requiring remote type of loading machines.

Due to abundance of locked up coal in thick seams of India, many experimental trails were conducted for extraction of full thickness of the seams to reduce exploitation losses and strata control problems associated with conventional system of depillaring (Singh, 1992). Other variants of pillar mining : Pocket and wing, rib pillar extraction, and Wongawalli system are also in use worldwide but is application was limited to normal thickness of upto 3 m (Singh, 1998).

Conventional system of depillaring in a single lift by caving in case of the seams developed along the floor invariably follow some principles in India, which can be summarized as i) The pillars split into stocks and extracted upto 3m height initially and on retreat upto 4.8 m irrespective of total thickness of the coal seam, leaving adequate tenders (ribs) along the goal side, ii) Minimum roof exposure upto 90 m^2 at any time. iii) Diagonal line of face advance to facilities strata control, and iv) the size of the panel such as to be completed within incubation period; commonly 6-9 months. In addition, for ensuring stability of the workings, splitting of the pillars was restricted to the two pillars ahead of the pillar under extraction (Rakesh and Prasad, 1995).

The recent experimental trails for extraction of full seam thickness upto 12 m include blasting gallery method and hydraulic mining. Of these, the former was successfully experimented in Godavary valley coalfields (Jayanthu & Singh et al, 1998) but the later method was proved to be not suitable for Indian mining conditions. The failure was attributed to comparatively hard nature of coal (Singh el at. 1992). Experimental trail of mechanized extraction of pillars in conjunction with cable bolts was proved to be successful in view of improved level of recovery from 40 to 75% productivity from 1.43 to 2.01 average monthly production level from 5,000 to 7,000 tons, and reduction in cost of production by India Rupees 80 per ton of coal (CMRI, 1997). Innovative methods of total seam extraction including developed pillars in seams of 15 m thickness were also proposed through suitable mechanization in Indian conditions (, Singh, R., et al, 2003)). On the other hand, highly productive proposition of longwall, oblique longwall or shortwall mining of developed pillars could not be experimented due to apprehension of complex strata control problems through numerical and equivalent material model studies (Singh, 1989). These methods, if successfully experimented to normal seam upto 3m thickness can be extended to thick seams with under winning mechanism as practiced in suitrage/integral caving in conjunction with longwall mining (Singh & Singh, 1999), but the later method was proved to be not suitable for Indian mining conditions. The failure was attributed to comparatively hard nature of coal (Singh et.al. 1992).

2.5.2 Extraction of Thick Seam (6.5m) by Continuous Miner:

Method of working adopted during Extraction of thick seam coal by Continuous Miner with diagonal slicing. The application of this method was adopted in NCPH Colliery R-6 Mine. The Seam thickness was 6.5 meters and there were no major geological disturbances in the Seam. There were two alternatives for the development of the Seam.

1. Development along the roof or top section of the Seam.
2. Development along bottom of the Seam i.e. bottom section of the seam.

In this mine there was usual practice of development of the Seam along bottom of the Seam. But this panel was developed along roof of the seam. The reasons behind this were -

- There was a dull coal layer in the top section (above 4 meters from the bottom of the seam).
- It was difficult to take out the roof coal at the time of depillaring if it was supported with rock bolt with resin capsules.

So the seam was developed along roof of the seam with as height of 4 meters leaving coal at the floor. After successful development of the seam it was considered for extraction of the pillar. There were two alternatives for depillaring the panel S-1. The two alternatives were –

1. Splitting and Fender method.
2. Diagonal Slicing.

1. Splitting and Fender method:

In this method splitting of pillar was done and then fenders 2, 3, 4,5,6,7 were taken out. In splitting the pillar additional time was spent for supporting the roof of split which would have to be cut twice. The roof bolter also needed to spend significant time within the pillar carrying out the support work.

2. Diagonal Slicing

In this method cutting the pillar was done in diagonal fashion from the original gallery without splitting. Details describe in chapter-3 as Field Investigation study report.

2.6 Strata Control vis-a-vis Continuous Miner Operation

The Bord & Pillar method is predominant method in underground in our country and our persons are more familiar with this method in comparison to other underground method. So it will be better to convert the Bord & Pillar working from semi-mechanized to fully mechanize. In this context continuous miner is one of the best suitable technologies for mass exploitation of underground coal deposits in our future coal demand of the country. Fully mechanized Bord & Pillar technology (without blasting working with Continuous Miner and Shuttle cars combination) is known as mass exploitation technology. Mass exploitation technology had already fair degree of successfully introduced at Anjan Hills, Beharabad, and Pinoura of S.E.C.L., Tansi mine of W.C.L. and recently practice of Jhanjara at E.C.L.

The result of this technology are encouraging production as well as productivity wise and also safety wise due to efficient strata management in underground coal mines. In underground mining strata management is one of the key contributor for improving productivity and safety aspects in mining operations. However continuous miner exposes less risk in roof strata due to cutting of coal as compare with blasting of coal. Standard continuous miner can extract coal a rate of up to 38 tons a minute depending upon the coal seam thickness and geo-mining conditions. Underground mining is impossible without proper strata control. Strata control assumes great importance-be it development or final extraction; be it Bord and Pillar or Room and Pillar or Long wall or any other underground mining method. Even today, fall of roof and sides is the biggest single cause of maximum numbers of fatalities due to any other single cause (see in table-2.2).

Table 2.2: Cause wise Fatal Accidents in Coal Mines, due to Strata Movement (As per DGMS Annual Report)

Year	Fall of roof	Fall of sides	Total	Total below-ground accidents	Percentage of accidents due to strata movement
1997	38	12	50	94	53
1998	35	15	50	80	62
1999	33	11	44	74	59.5
2000	27	14	41	62	66
2001	30	9	39	67	58
2002	23	11	34	48	70
2003	18	5	23	46	50
2004	26	8	34	49	69
2005	18	7	25	49	51
2006	13	4	17	44	40
2007	13	4	17	25	68
2008	13	7	20	33	60
2009	17	9	26	39	66.6
2010	15	6	22	41	53.6
2011	12	4	16	25	64
2012	9	5	14	28	50
Total	**340**	**131**	**472**	**804**	**58.64**

The roof fall and side fall accidents accounted for **58.64%** of all below ground fatal accidents in coal mines. This either shows neglect in taking proper strata control measures or shows improper understanding of forces that strata in vicinity of an excavation will not move in a manner that will cause damage to men and material and also that when required; it will come down in an excavated place in time and manner that mine operator desires.

Bord & Pillar or Room and Pillar method of underground coal mining is widely used in the USA, South Africa and Australia, with growing acceptance in Russia China and now exploring in Indian coal mining industry. The system does not have the same potential for bulk production as longwall, but is a very cost effective method of mining. The system is less complex than longwall, both to manage and operate and is far more flexible with a much lower capital cost. It is accepted that there are some geological conditions in India that lend themselves to longwall mining as opposed to room and pillar but there are very many mines that are very suitable for room and pillar. This system also allows the mechanized extraction of remnant pillars by a fully approved mechanized method of mining. Two fully mechanized systems was operated in India, at Chirimiri Mine in SECL and Tandsi Mine in WCL. Presently two more mine are in operation one at the VK7 mine of Singreni and others at the Jhanjra mine of ECL. Furthermore tenders are in the pipeline.

In case geo-mining conditions are favorable for adopting continuous miner technology in underground coal deposits excavation. It contributes major share in mass production, productivity, safety and economic viable in operation system with efficiency. With our experiences continuous miner can able to produce 1400 to 1600 tons per day depending upon the coal seam thickness and strata control technology. Continuous miners operate a room and pillar or bord and pillar mining method. A series of 4 to 6 m wide rooms are driven in the coal bed with pillars or columns of coal left standing to help support the roof. Roof bolts, typically 1.2 to 2.4 m long steel bolts, are inserted into holes bored into the roof to bind the strata together support the roof.

In association with continuous miner in underground coal excavation technology, more and more accurate predictions and assessment of ground behaviors are required. Thus monitoring roof strata deformation has become a fundamental approach to assess the risk of unacceptable rock response after excavation. On the other hand, systematic monitoring on deformation and load provides the basis to adjust and optimize excavation and support actual ground conditions. Field measurement provides the information on the global material properties of the surrounding rock mass, the stability of the opening structure, supporting system. It also increase the understanding of the rock mass deformation and the failure behavior around the opening after excavation and forms the links between the theory and engineering practices, in which, the observed behavior can be interpreted with the aid of computer simulation. The process of mine design (including support system) is a dynamic process that continuous during mining, which typically relies upon information gained by field measurement and monitoring. Generally, the purpose of underground strata monitoring in continuous miner operation mainly review as:-

- To assess the safety or stability of opening.
- To confirm the assumed or expected rock mass behavior.
- To improve the understanding of the fundamental rock mass behavior and failure processes.
- To obtain data for design and analysis.
- To provide data for numerical model verification.
- To confirm excavation performance predictions.
- To allow extrapolation for the prediction of long-term rock response.
- To provide quality control data.
- To supply data to assist in modifying and improving excavation design and development procedures, including remedial measures.
- To evaluate the effects of remedial work or changes in mining procedures.

During development, we want to prevent strata movement or permit it to a very, limited extent. This we achieve by putting supports. The excavated area is thus made safe for carrying on further mining activities. Initial stage development, excavations are narrow and in almost all cases, roof bolting can suffice, as coal bearing rocks are amenable to such as support system. Strata control during development is, thus simple. When it comes to final extraction, requirements of strata control are not that simple, and may vary from case to case and will also depend on the method of mining adopted.

As bord and pillar workings are developed roof support becomes a major consideration and operating cost item. Support requirements vary considerably and are dependent on the physical conditions in the seam, which are often depth dependent, and the length of time that the workings are required to be serviceable. The most common support problem in bord and pillar workings is at the intersections of headings and cutthroughs. The exposed roof spans are greatest at these points and therefore the intensity of roof support must also be greatest there if roof failures are to be avoided.

In difficult ground conditions where the workings' sole purpose is to develop main or auxiliary access roads for later extraction operations such as longwalls, a method of minimizing the intersection problem is to extend the length of the pillars to reduce the number of intersections and/or to stagger the cutthroughs to make all intersections three-way rather than full four-way holings. To ensure long term stability of the strata overlying the bord and pillar workings, if this is required, pillar dimensions are critical. The size of a stable pillar sufficient to resist creep or failure and to support the general strata above it is dependent on the strength and nature of the coal and surrounding strata, the height of the pillar, discontinuities such as cleats and mining induced fracture planes in the pillar, the pillar's width and length, the width of the headings defining its boundary and the three dimensional stress field existent at the pillar. Overseas, particularly in RSA, empirical formulae have been developed to estimate the size of stable pillars. It is not generally appropriate to apply these to Australian conditions because of differences in the many parameters listed above from seam to seam and district to district. If doubt exists about pillar stability then a full rock mechanics investigation and analysis for the particular site conditions is required.

In some Australian states legislation dictates the size of pillars and width of headings as a function of depth. This legislation is not firmly based on rock mechanics principles and may also be quite wrong. In very general terms it would appear that where headings are driven 6 m wide and no higher than 4 m, pillar centre dimensions of around 50 m x 50 m are usually stable to depths of 600 m in Australian practice.

During 1991-92 investigations took place through an Australian Mineral Industry Research Association (AMIRA) project to attempt to develop more appropriate criteria for the design of pillars. For the support of the roof immediately above the headings in bord and pillar workings legislation generally requires that some form of systematic support is to be installed. The intensity of the support varies from almost nothing in the case of shallow mines having hard conglomerate or massive sandstone roof to closely spaced steel cross supports or arches supplemented with bolts where the workings are deep and the roof is weak laminate, mudstone or coal. Timber supports both as props and cross bars, either full round or split, are still in use as a means of roof support but the trend to wider use of roof bolts, with or without steel straps is general.

The advantages of all-timber support lies largely in its yielding capacity which gives warning of convergence but its ultimate load carrying capacity is much less than that of steel supports or correctly designed roof bolting patterns. Additionally timber is becoming scarcer and more expensive to obtain. It is very bulky material to convey to and handle at the face and it is flammable and subject to fungal and insect attack. The extensive use of timber in main long term roadways inevitably involves on-going maintenance costs to replace defective supports and creates the possibility of widespread roof failures. As such it is rapidly falling from favour amongst operators.

Steel sections such as rolled steel beams, universal sections, reject rail sections, box beams and more elaborate pressed sections have been widely used as cross beam supports where roof conditions require more intensive support than timber alone can afford or bolting is ineffective. These are held to the roof either by timber props or steel legs and because of their weight they are usually provided with a safety bolt and saddle system to prevent them from falling should the props be dislodged accidentally. While they offer greater support than timber cross beams they are often very heavy to handle manually and increase roof support delay times and they do not always offer audible warning of impending failure. They are also more expensive than timber. Steel arches are used only in the most difficult of conditions for permanent roadways because of their cost and difficulty of erection. The actual cost of an arch and its associated parts may be up to Rs. 10,000/- and may take two hours or more to properly install so that support costs are very high and production rates are low when they are used. Moreover they are difficult to match with continuous miners which cut out for squared working face sections rather than profiled cross sections. Arches are better matched to boom type heading machines which have profiling capability and then only in critical headings. In general arches have no place in normal bord and pillar workings. With the improvement in roof bolting technology arising from speedier and more reliable bolting machines, either hand held or integrally mounted on continuous miners, and from resin anchoring in place of mechanical anchoring of the bolts there has been a general move towards the wider use of roof bolts as the main support for bord and pillar headings. Where the roof is friable or liable to weathering deterioration, the bolts are usually placed through light steel straps or perhaps steel mesh sheets to give a measure of cross support coverage. A popular bolt length in use is 1.8 m but depending on particular roof strata longer or shorter bolts may be more suitable for the conditions. Most bolting patterns have been derived by trial and error experiments but initial design guidance is possible from mathematical or physical modeling of the strata and working layout. The advantages of support by bolts are that it provides positive support and it is usually permanent. Quantities of support materials are much reduced, clearances and airway resistance characteristics of the resultant roadways are improved, and support time delays may be significantly reduced. There is usually no need to augment bolting patterns with side prop supports but timber props are sometimes retained to enable the erection of ventilation brattice and cables and to give warning of convergence, for the one defect with bolting is that little warning may be given if roof failure is imminent. Nevertheless roof support by means of bolt patterns only with no auxiliary timber props is becoming commonplace.

Accepting that the fundamentals of Strata control Techniques performances and potential of continuous miner will increasing trends in Indian coal mining sector. The results show at three Indian mines in terms of productivity, safety, and economical are comparable to global standards. The following table No. 2.3 shows that Continuous miner Performance in India.

Table- 2.3: Continuous miner Performance in India

Particulars	Chirimiri	Tandsi	Jhanjra
Date of starting	13.05. 02	01.09. 03	20.9.07
Highest production in a day, Te	3200	2400	2800
Highest production in a Month, Te	50,181	38,600	66,150
Production in Year, Te	463,297	3,14,705	631,770

The performance at both Chirimiri Tandsi and Jhanjara has been studied and the availability and utilization have been arrived at over the life of the projects. These results can then be compared with similar room and pillars figures from elsewhere in the World. The most striking statistic is that Chirimiri is already comparable with the average Australian room and pillar performance.

The Venkateshkhani underground mine (VK7 incline) of the Singareni Collieries Company Ltd (SCCL) in Kothagudem has achieved another distinguished milestone by registering an all-time high monthly coal production of about 1, 03,607 tonnes against its monthly target of 51,000 tonnes in January this year.VK7 incline, which is more than five decades old, has surpassed its previous highest monthly coal production record of 1, 00,700 tonnes achieved in January 1996, SCCL sources said. The sophisticated Continuous Miner Panel (CMP) of VK7 incline produced coal to an extent of 71,158 tonnes which accounts for a major chunk of the coal produced by the underground mine in the last month. The remarkable accomplishment helped the mine exceed the continuous miner technology-aided previous highest monthly coal production of 66,209 tonnes registered by the CMP of GDK 11A mine of the SCCL in Karimnagar district in year 2010-09, the sources said. In another significant development, the VK7 incline has produced 6.01 lakh tonnes of coal up to January as against the stipulated annual production target of 6.25 lakh tonnes in the 2010-11 financial years.

As per our past experience Chirimiri and Jhanjara mine has achieved exceptional performances with a potential of over 500000 tonnes per year. Tansi mine has difficult roof conditions and is not used to achieving high production and productivity. Even so , the difficult roof condition have controlled with modern " world best practice" roof bolting systems and output of up to 1200 tonnes per day has been achieved. It can further be emphasized that if this technology is implemented properly it will bring an unfathomable change in Indian mining scenario. There are many mines in India that may have potential and economic viability with Ground control Technology concepts for installing continuous miner systems and mass exploitation of underground coal deposits. This paper has shows our past experience and Ground control technology suitability of continuous miner system in present and future demand of coal with additional consideration of mass exploitation techniques.

Indian coalfields inherit difficult geo-mining conditions, which make it difficult to extract developed coal pillars by an underground mining approach. However, to meet the demand of coal production, three attempts of application of the mechanised depillaring have been made with mixed results. This paper, first, briefly presents results of the country's first fully mechanised depillaring face and, also, discusses the outcomes of two other attempts of mechanised depillaring. A brief review of the exiting geo-mining conditions Indian coal-fields shows that the role of support, both, natural and applied support is important issues, which need to be addressed for the varying conditions of different sites. The important technical point to notice during the first fully mechanised depillaring face is successful application of high capacity, pre-tensioned, stiff and resin grouted roof bolts as systematic support of roof (SSR) and breaker line support for the laminated roof of the site. A review of different continuous miner faces shows that the high capacity roof bolts and mobile roof support are complementary for weak and moderate roof strata. But, this approach of roof support may not always be applicable in India due to frequent encounter of massive roof strata. It is an open fact that pillar recovery at higher depth cover is another important emerging issue in the country. Coal mining activity at deeper cover is likely to intensify in the future because the shallower deposits are exhausting. At greater depths, the size of pillars is too wide to be fully extracted with single pass pillar stripping techniques. Here pillar splitting before extraction becomes the only alternative if the pillars are to be fully extracted. On the basis of experiences of different conventional semi-mechanised depillaring faces, this paper also discusses the rock mechanics aspect of manner of pillar extraction and efficacy of the left out fender for a successful high speed depillaring. It is realised that the experience of local rock mass behaviour is an important input for a successful adoption this mass production technology.

Future extraction of coal, mainly through underground mining, is going to encounter, relatively, deeper excavations in difficult rock mass conditions. In India, large number of coal seams has extensively been developed by formation of pillars to meet the increasing demand of coal in the country. This is found to be a simple and safe method of coal production from underground mines. The process of pillar formation also received favourable situation due to presence of the competent coal seam and similar surrounding rock mass. Indian Coal Mines Regulations (CMR), 1957 is also quite liberal for pillar formation. This strategy of coal production found suitable for the Indian coal mining industry because of low capital investment and involvement of trivial technical expertise. Now, industry is looking towards the huge amount of coal locked up in the pillars. Underground coal mining in India is facing serious techno-economical challenge during depillaring of the developed coal seams. Conventional depillaring frequently encounters strata control problem and the productivity remains quite low.

This Strata Control Technology for Mass Exploitation of Coal deposits with Continuous Miner is setting the pace for the Indian underground coal mining industry, which is intensely seeking modern technology to meet the ambitious growth targets of coal production for the Indian mining industry.

2.7 Strata Behaviour

Development or pillar extraction affect the magnitude of strata pressure and hence, its manifestation in convergence of advance workings and extraction area, stress over ribs, stooks and pillars during depillaring (Singh, 1998). Foreseeing the strata behaviour problems during depillaring in thick coal seams, Singh (1992) states "Strata movement of unprecedented nature are likely to occur with underground mining of thick seam. Strata control engineers will be required to constantly monitor these movements and review their concept of ground control so as to offer adequate solution to the problems that may arise and which are mostly site specific". Particularly when the thick seam working with caving operation, such critical conditions occur, and maximum pressure over ribs, stooks and pillars and convergence are more common observed during depillaring (CMRI, 1997; Singh et al: 2000). A proper understanding of the influence of strata characteristics on progressive convergence behavior and support performance is essential for reliable planning, rational support selection and thick seam coal mining by continuous miner. Strata and support behavior monitoring is required for understanding the performance of support system. Figure 2.8 illustrates the instrumentation required for strata monitoring in a typical development gallery of underground coal mine.

In South African collieries normally roadways are 6 to 7 m wide with continuous miner operation in board and pillar working. This dimension is chosen so as to allow maneuverability of appropriate mine machinery and equipment, to meet production requirements and to ensure roof stability. Over the last two decades, a great deal of research effort has been expended to understand the behavior of roadways in different strata within this width range. The results have improved the understanding of the strata behavior to such an extent that 6 m wide roadways are developed with a high degree of confidence. With the advancement of technology and the quest to mine at a lower cost, some mines in South Africa have started considering the benefits of other types of mine machinery that will require much wider roadways to ensure optimum productivity. The roadways required will have to be about 10 - 14 m wide. The challenge confronting these coal mining companies is the potential instability that might be associated with such wide roadways. This dilemma is aggravated by the fact that, apart from theory, nothing is known about the expected behavior of such wide roadways. This led to experimentation at a selected site in a South African coal mine where 10 m and 14 m wide roadways were developed and monitored. Earlier research highlights the prediction of elastic theory to reality and set the precedence towards understanding the behavior of wider roadways in South African coal mines. (Quaye, 2001)

To minimize the dangers from weighting on the pillar due to overhanging of roof in the goaf and to ensure that as small an area of un-collapsed roof as possible is allowed in the goaf, a suitable code of practice for induced blasting should be evolved in consultation with a scientific organization keeping in view the depth of induce shot holes being not less than 2.7 m, direction & spacing of shotholes, explosives used etc. so as to limit the rate of convergence [i.e., the ratio of C1/C2 is equal or less than 2, where C1 is daily convergence at a site in a day "n" and C2 is the average daily convergence at the site up to the previous day i.e. day (n-1)] and also to ensure complete filling of the goaf and release of any abutment pressures.

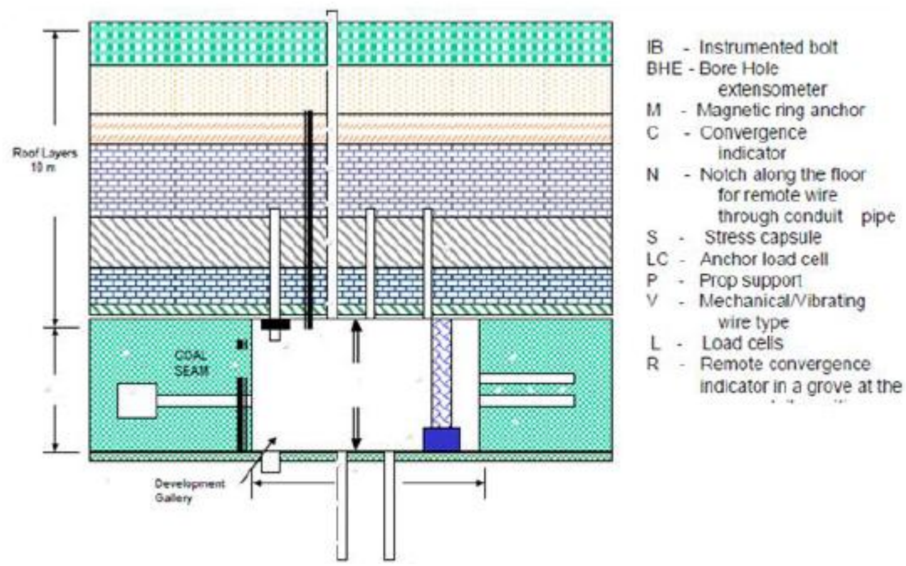


Figure-2.8: Typical Instruments for Strata Monitoring [Jayanthu et al 2008]

Convergence recording stations should be installed at all junctions situated within two pillar distance from pillar under extraction in the proposed panel. Monitoring of readings at convergence recording stations should be done in every shift by a competent person duly authorized by the manager and the measurements should be recorded in a bound paged book and the same should be counter signed daily by the Under Manager of the shift and Asst. Manager in charge. All the work persons should be withdrawn from the abutment zone if the ration of C1/C2 is equal to or more than 2 as given above and steps should be taken to release the goaf abutment pressure by induced blasting. The Safety Officer should co-ordinate recording, analysis and interpretation of the readings and advises the Officers/ Officials daily at the mine.

Study conducted regarding strata behavior with special reference to convergence and stress variation during an experimental trial of extraction of pillars with cable bolts as major support system in 6.5 - 8.0 m thick coal seam. Critical span for roof falls was estimated through empirical models and also evaluated by numerical models. In situ strata behavior studies during 1992-96 for about 70 local/major falls revealed inadequacy of the available literature for interpretation and prediction of the roof falls. Convergence acceleration and variation of induced stress based on continuous monitoring data showed distinct anomalies and potential for better understanding of strata mechanics during pillar extraction. (Jayanthu, 1998)

2.7.1 Convergence

Convergence is the manifestation of the forces which goes out of equilibrium due to underground excavation when the underground excavation are made, the sides of excavation tend to flow into the excavated area – the roof sags and the floor heaves. The net result is that the excavated area gets reduced. The distance between side and that between roof and floor decrease. The relative movement between roof and floor measured in the vertical plane is known as relative convergence or more commonly as convergence. Underground roadways suffer convergence during development and extraction, the former being essentially a logarithmic function of time (Unal, 1983). The convergence in Longwall workings follow more or less a predictable pattern, While it could be erratic in bord and pillar extraction because of the stocks and ribs, (Singh, 1998)

A number of tools have been described by (Kelly et al., 2002), but measurement of roof to floor convergence is a common practice in Indian coalfields for estimation of roof Strata behavior. On the basis of roof to floor convergence (C) recorded in and around the depillaring faces under hard and massive sandstone roof, Coal Mines Regulation (CMR, 1957) states following two criteria for anticipation of a roof fall:

$$\frac{C_1}{C_2} \geq 2.0 \quad \text{………………….…………..….} \ (2.1)$$

$$\frac{C_1 - C_1^{'}}{C_2} \geq 1.5 \quad \text{…………………………….} \ (2.2)$$

Where, C_1 is daily convergence at a site in a day n, C_2 is average daily convergence at the site up to the previous day i.e. up to day (n-1) and C'_1 is daily convergence on a day (prior to day n) when C1/C2 ratio is less than 1.5. It is also mentioned that the second i.e. eqn. (4.2) provides better result.

For the application of these two criteria, generally, roof to floor convergence measurement is undertaken at depillaring faces, applying simple telescopic rod (graduated) and putting pegs in floor and roof (Singh et al., 2004). This approach works for measurement in a development gallery but the measurement in and around the slicing is difficult by such a manual approach. In fact, the area around the slicing activity becomes inaccessible after withdrawal of the applied support. An electronic sensor based remote convergence indicators are used for such an unapproachable area. Combination of manual and remote measurements becomes necessary to know complete nature of the roof to floor convergence. One such measurement, conducted for development, splitting, slicing and after withdrawal of applied support at a depillaring panel (Singh et al., 2001) is shown in Figure 2.9. The change in roof to floor convergence remained very low in the presence of solid and big pillars. Here the measurement is done manually with the help of a simple mechanical type graduated telescopic rod till splitting of the pillar. The rate of roof to floor convergence increased with increase in percentage of extraction due to splitting and slicing. Slicing experienced relatively more change in the convergence and the area became vulnerable to experience roof/side falls. Under the situation, a remote convergence indicator was installed at the same point to continue the study. The connecting cable of the remote convergence indicator was taken out of the working to a safe place to get information even after the area became inaccessible. Withdrawal of support after slicing caused rapid increase in roof to floor convergence followed by roof fall. The sharp increase in roof to floor convergence at this juncture of the monitoring requires very high frequency of the observation. It is to be mentioned here that this complete observation of roof to floor convergence is obtained after placing twelve sets of instruments (each set consisted manual and remote measuring stations) in and around the depillaring face. An analysis of some convergence data (Satyanarayana et al., 2005; Mandal *et al.*, 2006) with the help of eqns. (1.1) and (1.2) showed dubious results than the indication of roof fall picked up by a complete cycle of observations as shown in Figure 2.9

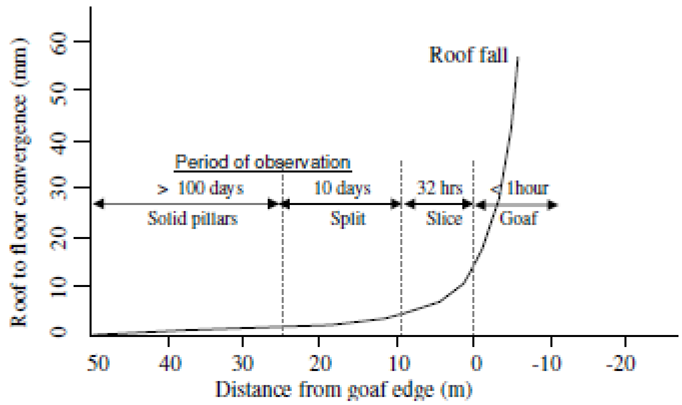


Figure -2.9: A complete set of roof to floor convergence observations in and around a depillaring face along with time period of the study.

Roof to floor convergence study works well for strata movement study during development of a coal seam (Ghosh and Ghose, 1992) but the operational constraints makes it difficult to be measured in and around a depillaring face. Further, for a stratified formation, there is a good chance that the roof to floor convergence may provide false indication of major strata equilibrium dynamics mainly due to movement of the immediate roof only (Figure 2.10). A recent strata movement study scheme (Shen et al., 2008) integrated three tools: roof deformation, mining induced stress change and seismicity for successful prediction of roof fall.

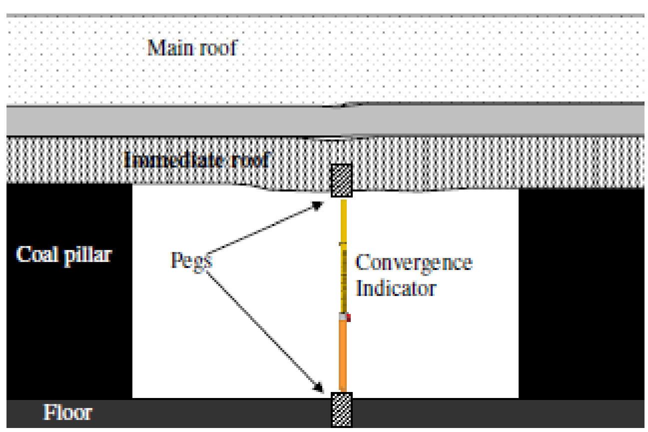


Figure- 2.10: An instrumentation scheme to study roof to floor convergence.

Many empirical relations were suggested for development galleries (Ozkan et al, 1997: Ghosh and Ghose, 1992) but a few attempts were made for depillaring workings in thick seams (NIRM, 1997) Trend of convergence (C in mm) with distance (D in m) from the line of extraction at the time of major roof fall for an instrument station during depillaring in 12 m thick seam was expressed by the following polynomial equation with coefficient of correlation 0.88

$$C = 10.7 - 0.66\ D + 0.01\ D^2 \text{ ---------------------------- } (2.3)$$

Singh (1989) conducted field studies Equivalent material model studies on convergence during depillaring 4.6 m thick upper Kajora seam with 4.6 m and 5 m working height the model, respectively at 60 m depth cover Field observation was incomplete due to disturbance of the stations while winning roof and floor coal. However, the model findings include: i) 51 mm convergence when the ribs were reduced to 1 m, ii) 160 mm convergence when alternate ribs were knocked down, and iii) roof with convergence upto 6.16% of the working height remained hanging, which followed sudden roof fall with 65 m span in the goaf.

In blasting gallery method of extraction of pillars to total seam thickness of 7.5 m, the rate of convergence increased about 4 days before first major fall (CMRI, 1987), This continuous increase in convergence was 1- 1.5 mm/day. During subsequent major falls, rate of convergence within 10 m of the extraction line was 2.5 mm/day, and beyond 10 m, it remained within 1.5 mm/day.

Shepherd and Lewandowski (1998) conducted convergence monitoring during wongawalli system of pillar extraction in a 2.5 m thick seam. Roof and floor closure next to the break line was measured using telescopic convergence rod device at the goaf edge. This indicated 120 mm total convergence during which 100 mm occurred within 50 minutes of goaf fall event.

Naismith and Pakalnis (1982) monitored coal pillar extraction in 3.5 m thick seam at a depth cover in the range of 25-120 m mining was a conventional cut – drill-blast operation with splitting and slicing. 3 mm convergence was measured three days before occurrence of major fall without any perceptible acceleration. However, on a day prior to the fall, a Very little pillar noise was recorded with slower acceleration rates. Roof cracks appeared on the goaf side about 6 hours prior to the fall with accelerated convergence. They recommended for convergence observations on round the clock basis.

Shephered and Chaturvedula (1992) monitored convergence during wongawalli system of pillar extraction with remotely read telescopic convergence rods to measure roof the floor closure between the completion of a particular lift in a fender and the occurrence of goaf fall. Fall warning time was noticed as 10 minutes at goaf edge while critical convergence was 70 mm.

As per literature of Jayanthu (2005) found that increase in roof deflection during pillar extraction for a seam of 2 m thickness at 260 m depth cover with convergence measurements using surveying leveling instruments. Roof movement of upto 15 mm was reported with 25 days as a function of time during pillar extraction.

Follington and Hutchinson (1993) and Follington et al (1992) presented the results of application of continuous monitoring for investigation of rock mass response to mechanized pillar extraction. The rate of change of loading and convergence was found to be useful to identify the onset of instability in the lift area of wongawalli system of pillar extraction at Lelahan No. 1 colliery in Queensland. Convergence upto 95 mm was recorded at different stages of extraction of lifts indication scope of providing warning signal.

In board and pillar extraction, roof fall is an important criterion for caving method of depillaring. Attempts were also made prediction of major roof falls during depillaring with a contention that measurement of roof to floor convergence is probably the most convenient method of insitu monitoring of rock behaviour and can afford a simple means to predict roof falls when extraction is in progress (CMRI, 1987). Four methods of prediction on the basis of cumulative and rate of convergence per day were applied but the probability Yield zones and its effect were also considered due to introduction of singularity. In India, seismic monitoring system is not well established but stress (vertical mining induced stress) and roof deformation measurements are being widely practiced. It is observed that, for strong and massive roof strata, monitoring of mining induced stress (vertical) provides rather better results (Shephered and Lewandowski, 1998; Singh *et al.*, 2004) than the roof deformation study.

2.8 Roof Support Design Methods in Mechanized Board and Pillar for wider entry (6m)

Various methods for the design of roof support systems have been proposed through the last 20 years with the ultimate objective of maintaining safe conditions in underground mining environments. The effectiveness of roof support systems is intimately linked to the geotechnical behavior and geologic makeup of the rock mass comprising the roof. Detailed geotechnical investigations are therefore, important to design effective roof support systems.

Empirical and analytical tools round out the primary design references which were investigated to determine the most effective roof support system for the different geomining conditions.

In this reference literature study and found, two categories of roof support design concept mostly acceptable in standardization of roof bolt support system for in mechanized Board and pillar mining mass exploitation of coal.

a) Dead-Weight Design

The concept of dead-weight design was proposed by Obert and Devall (1967). It is a limit equilibrium method wherein dividing the sum of the forces promoting the event by the sum of the forces opposing the event produces a safety factor. The following equation, developed by Obert and Duvall, can be used to determine the required bolt capacity to support a dead weight slab of roof rock:

$$P = \left[\frac{U \times t \times W_e \times R}{n + l} \right] SF \qquad \text{………………………….. (2.4)}$$

Where: P = required bolt capacity

U = unit weight of the rock;

t = thickness of suspended rock;

n = number of bolts per row;

W_e = entry width;

R = row spacing; and

SF = safety factor.

Equation 1 predicts bolt capacities of 7.5 and 11.2 tonnes for support patterns of five bolts per row on 1.2-meter centers and four bolts per row on 1.5-meter centers in sandstone (U = 2.6 t/m^3, t = 1.6 meters) for a roadway of 6-meter width respectively. Both standard rebar and mechanical bolts are relevant to this analysis.

A more commonly assessed application of dead-weight design is wedge support analysis. Modes of structurally controlled failure can be analyzed by the means of the stereographic projections technique, described in detail by Goodman (1989).

The results indicate 1.8-meter point-anchored #6 resin rebar provide adequate support capacity for third- and half-span pyramid geometries in 4-way intersections of 6-meter square dimensions. Anchorage depth and bond strength provided by pattern bolting was completely ineffective for the half-span height hemisphere geometry and partially effective for the third-span height dome geometry.

b.) Rock Load Height Design

The rock load height concept is a slightly more sophisticated version of the deadweight theory. Originally proposed Terzaghi (1946), the theory predicts the load on the supports based on the rock quality and by the roof span. Unal (1983) defined the rock load height for coal mining:

$$h_t = \left(\frac{100 - RMR}{100}\right) w_e \qquad \text{……………. (2.5)}$$

Where,

h_t = Rock load height, m

RMR = Rock Mass Rating

w_e = Entry width, m

This equation implies a maximum rock-load height equal to span. It can further be shown that the rock load is proportional to the cube of the span. When the geology is not uniform, and roof falls are truncated by an overlying self-supporting strong bed, the rock load height may be constant regardless of the intersection span. In this case, the rock load increases in proportion with the square of the span (Mark & Barczak, 2000).

Unal developed equations for support design based upon his research of coal mine roof support in the US. The equations were summarized as design charts for 6-meter wide entry and corresponding 4-way intersections. The governing equations are listed below.

For point-anchor bolts:

$$S = \frac{\sqrt{C}}{SF \times \lambda \times h_t},\ L = \frac{h_t}{2}$$

For full-column resin bolts:

$$S = \frac{\sqrt{C}}{SF \times \lambda \times h_f},\ L = \frac{\sqrt{B^2 \times h_t}}{300} \qquad \text{…. (2.6)}$$

Where: γ = unit weight (lbs/ft^3),

C = bolt capacity (lb),

S = bolt spacing (ft)

B = span (ft),

L = bolt length (ft),

SF = safety factor,

h_t = rock load height (ft)

Unal's also suggested roof support guidelines for entries and 4-way junctions in the continuous miner operation.

2.9 Designed Support System in Study Area

To fulfill the objective of study, as a field investigation were carried out based on field data collected from NCPH Colliery R-6 Mine. We have also verifying with numerical modeling spread over the major load in the support system. After data collection from mine, numerical modeling was conducted for verifying and modification. All relevant information such as plans, sections, RMR (where available) was collected along with roof rock behavior and support performance where needed from the mine. For numerical modeling, parametric changes in the following factors were made for the mine:

i) horizontal to vertical in situ stress ratio, K

ii) width of split/slice, W_{sp} or W_{sl}

iii) rock mass rating, RMR

This gave rise to various models with as many output data sets. An example of safety factor contours plot (3D numerical model result) in and around a slice junction and slice during simulation of a depillaring panel S-1 at NCPH Colliery R-6 of SECL at the time of main fall. Regressions were done to give separate equations for **Rock Load Height (RLH)** and required **Support Load Density (SLD)** for the slice, junction, split and goaf edge.

The developed equations are as follows:

For slice junction, $$SLD_{jn} = \frac{\gamma . H^{0.50} . K^{0.64} . W^{1.17}}{R^{0.90}} \quad \ldots\ldots\ldots\ldots\ldots (2.7)$$

Within slice, $$SLD_{sl} = \frac{\gamma . H^{0.67} . K^{0.84} . W^{1.74}}{R^{1.42}} \quad \ldots\ldots\ldots\ldots\ldots\ldots (2.8)$$

In the split gallery, $$SLD_{sp} = \frac{\gamma . H^{0.52} . K^{0.59} . W^{1.12}}{R^{1.02}} \quad \ldots\ldots\ldots\ldots\ldots (2.9)$$

For goaf edge, $$SLD_{ge} = \frac{\gamma \, . \, H^{0.54} \; K^{0.49} \; . \; W^{0.89}}{R^{0.79}} \quad \text{.................. (2.10)}$$

where, γ is the weighted average rock density of the immediate roof strata, t/m^3, H is depth, m, K is the ratio of horizontal to vertical in situ stress, W is the width of split or slice, m and R is the weighted average RMR of the immediate roof rock. SLD_{jn}, SLD_{sl}, SLD_{sp} and SLD_{ge} are the required support density in t/m^2 at the slice junction, within slice, in the split gallery and at the goaf edge respectively.

To supplement the modeling results, NCPH Colliery R-6 Mine was used for the field instrumentation where instrumented rock bolts were used to determine the axial load, bending moment etc developed along the bolts along with stress meter observations for ribs stability. Field results are very much close to the modeling results for a mine. After going through the developed equations (Eq. 2.7 to Eq.2.10), it is clear that five variables are to be known to estimate the required Support Load Density at different places of the face during depillaring operation. These variables are the **depth of cover H**, **in situ stress ratio K**, **Rock Mass Ratings** of the immediate roof rock **R**, **split and slice width W** and **rock density γ**. Estimation procedure for these variables are described below except for **H and W**, which are directly obtainable. Measurement is the best method to determine the ratio of in situ horizontal stresses to vertical stress K for any particular mine. In the absence of the *in situ* measurements of stress values, theoretical values can be used. Theoretical value and measured *in situ* stress worldwide showed that the in situ horizontal stress does depend on elastic constants (Sheorey, 1994) of the rock as given below:

$$\sigma_h = \frac{\nu}{1-\nu}\sigma_v + \frac{\beta E G}{1-\nu}\left(H + 1000\right) \quad \text{.. (2.11)}$$

Table -2.4: NCPH Colliery R-6 Mine for 3-D numerical modeling

Name of the colliery , seam & panel	1*- Depth, m; 2* - Dev. gallery size, m; 3* - Pillar size (c-c), m; 4*- Split width, m; 5*- Slice width, m; 6*- Extr. height, m						Weighted average roof rock properties			
	1*	2*	3*	4*	5*	6*	σ_{c}, MPa	σ_{t}, MPa	b	RMR
NCPH Colliery R-6 Mine, No. 3, S-1	67	6x4	22x22	6	6	4	29	2	0.5	48.07

Where, σ_h and σ_v are the horizontal and vertical in situ stress respectively in MPa

υ is the Poisson's ratio = 0.25

β is the coefficient of thermal expansion, / ^{0}C

E is the Young's modulus of the rock, MPa

G is the thermal gradient = 0.03 ^{0}C/m for coal measure rocks

H is the depth of cover, m

After putting the value of υ and G, the above *in situ* horizontal stress will transform as:

$$\sigma_h = \frac{\sigma_v}{3} + \frac{\beta E}{25}(H + 1000) \qquad \text{...(2.12)}$$

The in situ vertical stress can be written as:

$$\sigma_v = \gamma H = 0.025 H \qquad \text{... (2.13)}$$

The value of β for coal can be taken as 30 x 10^{-6} / ^{0}C while for other type of the coal measure rocks 8 x 10^{-6} / ^{0}C (Sheorey et al. 2001). On the other hand, Young's modulus of each type of the rock lying in the roof strata up to the height equal to gallery width can be tested in the laboratory and its weighted average can be estimated. The value of β should also be a weighted average. The value of RMR, which again should be weighted average, can be readily determined from the CMRI classification tables. If the Rock Mass Rating (RMR) of any layer lying between immediate roof rocks of height equivalent to gallery width is more or equal to 70, it should be ignored during estimation of weighted average of RMR. The density γ can be measured by standard lab method and should also be a weighted average. Once we know the required support load density in the split gallery, at the slice junction, within the slice and goaf edge of the depillaring face, the selection of proper support systems as per requirement (Continuous Miner operation) and support pattern for respective areas can be designed. Based on past experiences on roof supports and their effectiveness along with pull test of different types of the bolts used in different underground mines, the load bearing capacities of different type of Roof bolts support items are given below in table 2.5, (CMRI Report, 1987).

Table- 2.5: Load bearing capacity of Roof bolts support systems

	Type of Roof bolt Support	Load bearing capacity (t)
1	Roof bolt (full column grouted with quick setting cement capsules) (TMT ribbed bolt of 22 mm dia)	6
2	Roof bolt (fully column grouted with resin capsule) (TMT ribbed bolt of 22 mm dia)	12

To estimate the applied support load density by different support system used in the mine, Eq. 2.14 can be used as (Sheorey et al., 1987):

$$Applied\ \ Support\ \ Load\ (ASL) = \frac{n\ .\ A + m\ \ .\ Q}{W\ .\ \ a} \quad \text{T/m}^2 \quad \text{.......................} \quad (2.14)$$

Where, n is the number of bolts in a row,

A is the anchorage strength of each bolt in **ton**

Q is the load bearing capacity of the additional support if done in **ton**

m is the number of additional support at a spacing "**a**", if it has been used

W is the width of split or slice in m

a is the spacing between two consecutive rows in m.

Once we know the magnitude of applied support load density, we can easily determine the safety factor of support system using the equation:

Safety factor of supports = Applied Support Load / Support Load Density = *ASL / SLD*

While using the equations for *SLD* developed in this report, it should be realized that they have a built-in safety factor of 1.5. As such $ASL / SLD \geq 1.0$

2.9.1 Strategy for the selection of the rock bolt support system

The purpose of Rock bolt support is to provide a safe working environment while allowing economic extraction of the coal resource. Rock bolt systems are designed essentially based on considering the stress field around the opening and analyzing the structure of the rock mass. The stress field should be considered by magnitude and direction as well as its influence on the rock mass. The change of stresses in the rock mass depends on a number of factors such as excavation geometry, rock mass condition and inherent rock stress situation. Whereas close to the surface rock mass conditions are most important, with ongoing depth the ratio between the induced stresses and the strength of the rock mass become more and more important for the stability of excavation. There are some methods available to estimate the stability of openings through that relationship when the opening is unsupported. Rock bolts are an intrinsic type of roof support installed into the roof. Roof bolts are loaded as the roof deforms, and they interact with the rock to reduce bed separation by confinement. The selection of rock bolts available to an underground coal mine management is unpredicted. During the design of a rock bolt system a lot of factors have to be considered. The most important factor is rock mass characteristic including geological and hydrological conditions. The design of roof bolt systems can be performed by:

- Analytical solutions,
- Empirical assessments,
- Numerical modeling.

The existing rock bolt support system has been analyzed in this study in following two way assessments:

- Field assessments,
- Numerical modeling.

Several design procedures have been developed for rock bolts. Generally, these procedures relate the characteristics of the rock mass, the stress conditions and the opening geometry to each other and allow developing a proposal for support requirements. The main objective of installing roof bolts in underground excavations is to minimize the deformation that induced in the immediate roof as well as to redistribute the stress in the rock surrounding the opening (Stillborg, 1986). To achieve that, the mechanical behavior of rock masses reinforced by the fully grouted bolts, i.e. rock-bolt interaction needs to be fully understood. The existing support of roof bolts has been verified into two ways in this study.

2.9.1.1 Field assessments

Most field assessments involve measuring the force profile along the bolt so that a complete load transfer history during different mining stages is determined and the performance of the roof bolts can be evaluated. The pull out test and the instrumented bolt are two main tools to measure changes in bolt load and examine rock-support interaction in the field.

Pull out test - The pull out test is a popular technique for testing the resistance of the bolt to axially applied downward load. The pull out test is considered a valuable test for the fully-grouted roof bolt to test the bond strength between bolt, resin and hole. The tested bolt is installed in the same way and by the same material as its intended use. The bolt is pulled out hydraulically and the displacement at the bolt head is measured at each pre-determined step of load. The bolt is pulled until the bond with the rock fails. From a plot of load verses displacement the ultimate capacity for the bolt can be determined (Figure 2.11).

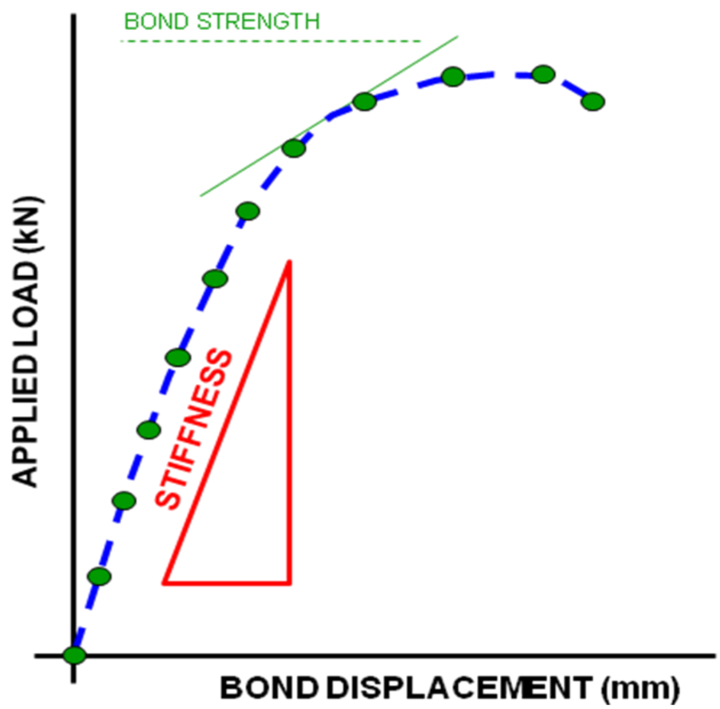


Figure- 2.11: Typical pull-out test results showing "yield point"

Franklin and Woodfield (1971) conducted pull-out tests on polyester resin anchors installed on different rock types with different strengths such as granite, coal, limestone, chalk… etc. They developed the following relationship between the required bond length and rock type, so that, the strength of resin anchor could be adjusted to suit the rock strength by using varying amount of resin:

BL = (BF x L) + SF…………… (2.15)

Where BL is the length of bond needed to give less than 5% anchor failures at the design load; L is the required anchor strength; SF is the safety factor, 6 inches in strong rock and 12 inches in weak rock; and BF is the bond factor that can be obtained from Figure 2.12.

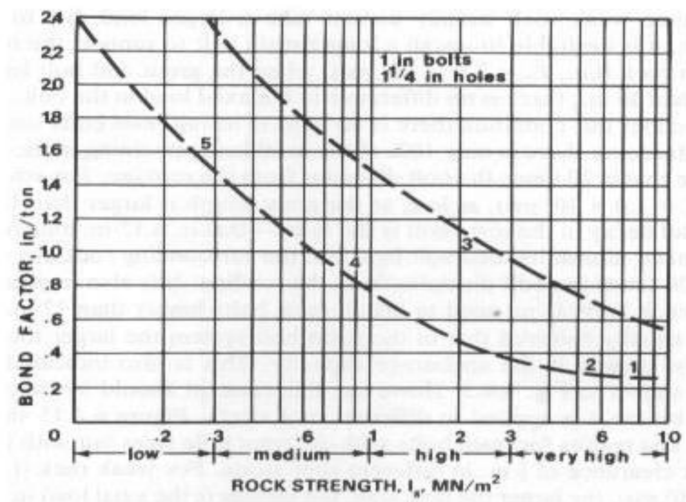


Figure- 2.12: Bond Factor versus Rock Strength, (Franklin and Woodsfield, 1971)

If the total stress (from field measurements) is greater than the maximum allowable stress, either one or a combination of the following three methods can be used to reduce the stress in the bolt:

1. Reducing the bolt spacing,
2. Increasing bolts density, or
3. Increasing the diameter of the bolts.

Disadvantages of using the instrumented bolt method in design - Although the instrumented bolt can provide us with details about axial and bending load distribution along the bolt length, it has some disadvantages (Signer et al, 1993).

1. The rebar is milled with a certain depth along each side, which will cause incorrect representation for the bolt area.
2. Maximum axial load or bending moment may be reached between the locations of the strain gauges and is not been measured.
3. The alignment of the strain gauges is critical to obtain good results.
4. The failure of strain gauges in some locations could be a result of wire failure or excessive loading, and can cause missing one or more of the axial loading values (Signer and Lewis, 1998).

Field test methods, installation and monitoring, are time consuming, very costly and complex (Guo & Stankus, 1997). Moreover, they seldom provide enough information for determining the stability of the whole entry system (roof, floor, ribs, bolts…etc.).

2.9.1.2 Numerical Modeling

The analytical tools are not sufficient to analyze the complex interaction between the roof strata and fully grouted bolts. Therefore, numerical models will be employed to investigate the stability and behavior of underground opening. In recent years, many numerical models have been developed for that purpose. These numerical models include finite elements, discrete elements, boundary elements and finite difference codes to solve a given problem. A large number of variables and complex site-specific conditions such as rock properties, geology, bolt type, bolt length, bolting pattern, mine geometries, etc., can be considered in the modeling. The limitation for the numerical modeling is the difficulties in determining in-situ rock properties and rock behavior as input parameters. With the development of powerful computer and numerical techniques, many commercial numerical packages are available, among which the ones using the finite element method such as FLAC (Fast Lagrangian Analysis of Continua) and UDEC (Universal Distinct Element Code), are most popular.

Stankus and Peng (1996) proposed the Optimum Beaming Effect. This concept has been used in practice and achieved some effect in the entry support in coal mines. Peng and Guo (1989) developed a 2D boundary element model for the design of fully grouted bolting. The model considered weak bedding planes, roof stiffness, layer thickness and horizontal stress. By applying dimensional analysis, a series of equations were derived to determine the bolt length, the number of bolts required to prevent bed separation, tensile fracture, shear fracture at mid-span and shear fracture at the entry corners.

The main objective of this study is to verify the suitability of grouted bolts for ensuring safety based on field observations as well as numerical simulation model. The main tool to achieve this objective is the commercial finite element program FLAC version 5.8. FLAC is a command-driven program, requiring the user to provide a series of commands to control the operation of the program. Figure 2.13 is the flowchart of the process for building a FLAC model. In order to set up a model, three fundamental components of a problem must be specified:

- ❖ A finite difference grid, which defines the geometry of the problem;

- ❖ Constitutive behavior and material properties, which dictate the type of response the model will display upon disturbance such as excavation; and
- ❖ Boundary and initial conditions, which define the in-situ state; that is, the conditions before a change or disturbance in the problem is introduced.

After these conditions are defined, the initial equilibrium state is calculated for the model. Once the model responds as expected, alterations, such as making an excavation and putting bolts in the roof are made, and the resulting model response is determined.

FLAC uses an explicit time-marching method to solve a problem. The solution is reached after a series of computation steps called cycle steps. The history of mid-span vertical displacement and displacing speed is monitored to see whether the opening is stable or plastic movement occurs. A sharp increase in the magnitude of displacement is an indicator of plastic movement. By contrast, if the displacing speed converges toward zero or the vertical displacement settles down to a certain magnitude, the immediate roof of the opening then reaches its stable state.

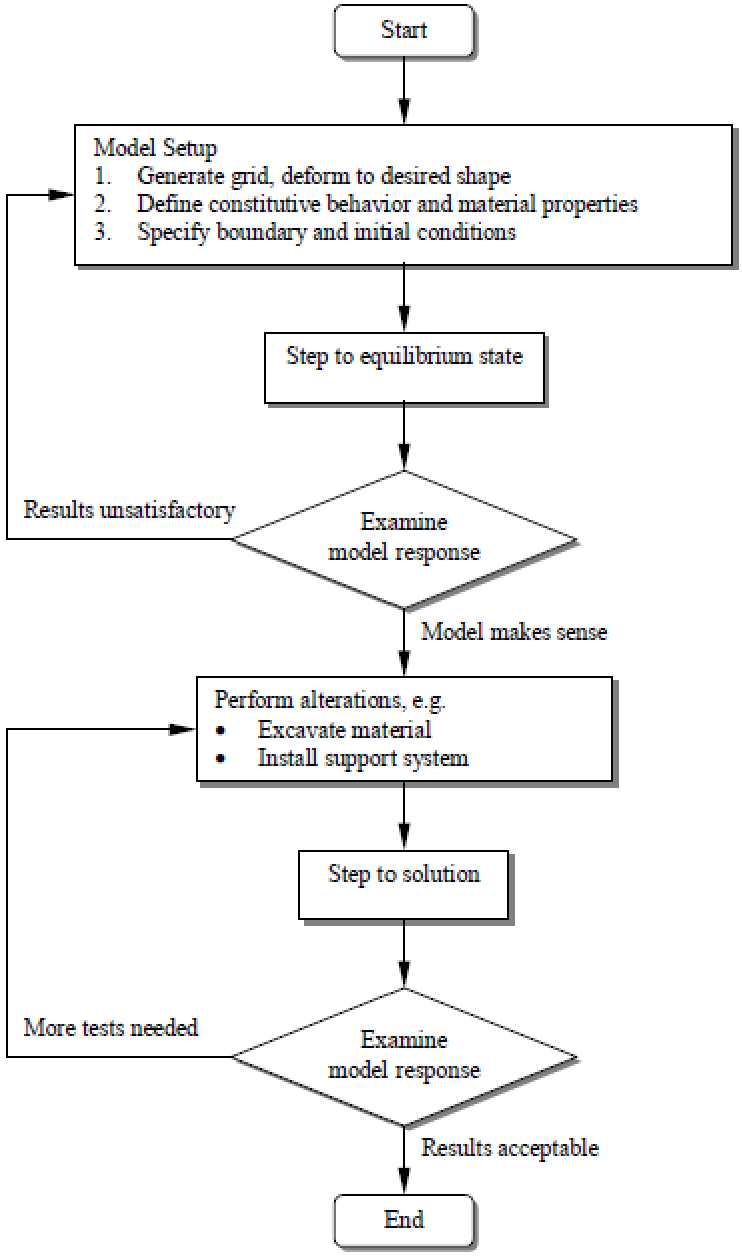


Figure- 2.13: General steps of model building and problem solving in FLAC

The problem being modeled is broken down into two phases. Phase 1 includes grid generation, specification of constitutive behavior and material properties, definition of boundary and initial condition, computation of equilibrium state, and excavation of the opening. Phase 2 includes installation of bolts and problem-solving. This program was used to analyze and model the effectiveness of fully grouted bolts in the stability of roof strata and develop a design methodology and guidelines.

Using FLAC 5.8, finite element models were developed to simulate the components of the fully grouted bolt and its interaction with the surrounding roof strata. Several models were conducted in order to investigate the effect of the fully grouted bolt on roof stability, and the load transfer from the rock to the bolt and vice versa under different geological, mining conditions and bolt parameters. The behavior of the simulated fully grouted bolt was verified with a case study during the development and mining stages. Different failure modes of the fully grouted bolts, such as bolt axial failure, bolt shear failure, and grout/rock interface shear failure, were defined. Bolt and roof stability measures were presented in order to evaluate the bolt and roof stability.

Investigation of trends using numerical modeling for comparing of empirical model of support

As mentioned earlier the behavior of the roof is a function of many variables. These include the stress environment, roof lithology and strength of roof materials, board width, etc. A further complication is that the variables govern the roof behavior according to their combination with the others. There are a great number of different combinations, and although great care was taken to include the widest possible range of parameters in the study, it was clearly not possible to include all or even a sufficient number to derive all the answers experimentally. However, important trends were derived. To investigate these further, a numerical modeling code was added to the research program. The three dimensional boundary element codes FLAC 3D was used in the analysis. The basic three dimensional models that were used are shown in Figure 2.14

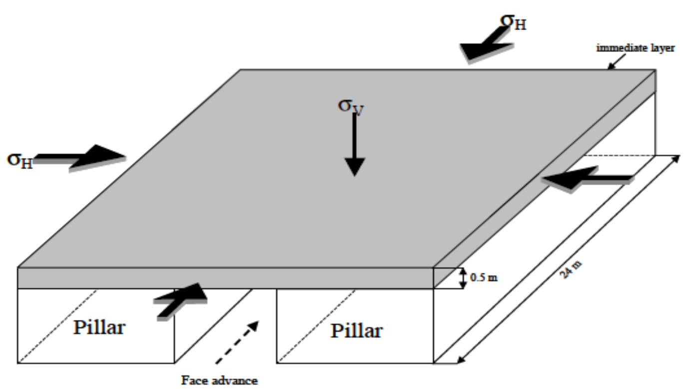


Figure- 2.14: The basic FLAC 3D Model that was used in the numerical modeling analysis

2.10 Remarks on Previous Studies

The literature review presented in the previous sections indicates that strata behavior with respect to convergence and several types of roof bolts support are currently being used in coal mines. But no reliable design approaches are available for coal mine wider entry (6 m) support and thick seam extraction by continuous miner although various methods have been tried to understand the support mechanisms and to produce rational design guidelines.

The inadequacies of the previous studies can be summarized as follows:

1) The rock properties are not known from in-situ conditions and specific sites, so any calculation and simulation poses errors.
2) Previous tensioned bolting design did not incorporate bedding plane effects, which is very important for the stability of the roof.
3) Various types of roof with different stress level have different failure modes and the bolt support should prevent possible failure, but none of the available design methods takes failure modes into consideration.
4) Convergence contributes a lot to roof falls, but few methods use convergence as a factor in verifying stability of roof bolting.
5) Most of the numerical models used in simulating bolts are elastic and two dimensional, which is not able to represent the actual rock mass response and bolt-rock interaction.

This study will try to consider the roof failure modes and verifying existing roof bolting support by using 3-D numerical method.

2.11 Justification and Significance of Present Study

The main purpose of the study are strata stability analysis to provide safe and productive design of the method of work with more space for implementation of large scale mechanization to meet ever – increasing demand of coal in the country. These can be fulfilling with the implementation of the following outcome of the present study are:

A. Design of suitable support system through monitoring of strata behavior & modeling.
B. Setting of Standard guidelines for Mass Exploitation of underground coal deposits are yet to be deploy in develop model instead of present mining methods.
C. Possible changing added in existing legislative provisions which are not mach with the mass exploitation technology.

The progress of the technology in many branches of engineering is quite rapid in recent years. However, in case of underground coal mining, the progress is not as expected as to be. It remained a lot with traditional systems, and only a few attempts were made to adopt recent trends. Although it could be attributed partly to availability and adoptability of the modern mining machinery, but also due to limitations of available strata control technology (Singh, 1989). In Indian coalfields, general practice to control strata is supporting the excavated area by suitable, efficient and necessary means. In normal practice excavated galleries are supported by some means without completely studying and analyzing the behavior of rock causing fall of roof and side in underground coal mines in India. Strata control is a major problem, which affects safety and productivity in underground mining. Roof fall is a cause of uncontrolled strata. The primary causal factor for poor roof is presence of week bedding planes. Geological disturbances such as joints, slips and faults, rank second in importance. Shale in the roof of coal seam is responsible for deteriorating roof condition primarily due to weathering of the type of rock in contact with water or humid mine atmosphere (Banerjee et al, 2007). Strata Control deals with the adaptation of a system by which control on the strata movement can be advised to a desired level to make workings safe and extraction of mineral possible. (Mathur, 2007)

Underground excavations in rock cause redistribution of stress around the opening. Depending on the strength and deformation behavior, the rock adjusts itself by moving into the opening. Sedimentary rocks have low tensile strength normal to the bedding plane, and low shear strength along the bedding planes. Adverse geological conditions in any area may further reduce the overall strength of rock mass. The deformation of roof has added advantages having gravitational forces, hence are liable to failure. Timely and proper support is necessary to prevent collapse of roof. (Wagner, 1974)

2.12 Summary

In the literature review, it is found that the concept of strata control came in existence since introduction of safety factor approach for stability of excavation during underground mining of coal. Introduction of scientific approaches for sustainable growth of underground mining industry in different parts of the globe changed this basic concept into different theories and models (Jeremic, 1985; Alan et al., 1987; Mark, 1992; Heasley 2000; Jayanthu et al., 2004). However, it is still a problem to accurately assess the stability of excavation area and support behavior under varying geo-mining conditions of different coalfields and efforts are on (Verma and Deb, 2008; Poulsen, 2010) to smarten the scientific approaches for a better result. Even the dimension of application of this parameter has widened and, today, this parameter is useful not only for natural support design but for applied support design, coal mine thick seam exploitation and evaluation of performance of mining methods.

Researchers have attempted to provide qualitative and quantitative information about strata behavior with respect to convergence during wider entry (6 m) and extraction of coal in a thick seam (6.5 m) by continuous miner with diagonal slicing. The selected topic for this research covers to establishing relation between development of convergence during wider entry for deployment of continuous miner as well as verification of support performance during pillar extraction in thick seam working. There are various parameters which affects development of mining induced stress. Details about the effect of these parameters studied by different researchers are mentioned above. However, the parameters influencing significantly in convergence phenomena during extraction of coal seams are depth of cover, characteristics of overlying roof formations, mining operational parameters and in situ stress conditions of the site. Some other parameters like geological features (fault, fold etc.) present in the working panel and characteristics of goaf materials also affect the mining induced stress development. It is an observed fact that the geological origin of rock mass affected development of a uniform scientific theory, even for the mining method.

During the literature review it is found that most of the theoretical and empirical models developed by different researchers are valid for the condition of the respective coalfields. It is also found that the formulations relationships/models to estimate nature and amount of mining induced stress varies considerably, mainly, due to change in characteristics of the site conditions and geology. Further, most of the models are developed for load estimation during development stage only and for longwall workings, and the case of depillaring is, relatively, less emphasised.

There are some studies reported for Indian coalfields but not related to continuous miner working. (Singh, 1989, Ghose 1992, Jayanthu 1998) Many empirical formulations has suggested for development of gallery although not related to extraction of coal in a thick seam (6.5 m) by continuous miner with diagonal slicing. Jayanthu et al. (2004) also conducted field and laboratory investigations considering geo-mining conditions of Indian coal mines to assess the maximum convergence and vertical stress over rib/stook during depillaring. He has observed influence of height of depillaring over the amount of mining induced stress and showed that the stress decreases with increase in working height during depillaring. Again, the conducted field study remained limited to one mine and more emphasis was given to study the influence of height of working than development of a general approach to assess nature and amount of mining induced stress for a given site conditions.

The literature review clearly reveals that depth of cover, characteristics of overlying roof formations, mining operational parameters and in situ stress conditions are the main parameters which affect significantly in development of mining induced stress. This study also indicates that a model/theory to assess convergence development suit local geo-mining conditions and rock mass behaviour. As per above reported review, there is a need to identify strata behavior in different geo-mining parameters affecting in development of convergence during depillaring of thick coal seams by continuous miner with diagonal slicing and development of empirical models for verification of support behavior.

CHAPTER-3

FIELD INVESTIGATION

FIELD INVESTIGATION

3.0 General

Comprehensive field investigation of strata behavior with respect to convergence and monitoring of roof bolts support performance have been conducted at NCPH Colliery R-6 Mine of SECL during development and also pillar extraction of coal in a thick seam (6.5 m) by continuous miner with diagonal slicing. The response of the roof strata has monitor in gallery widening 4.5 m to 6 m in earlier development in Board & pillar method of working and also extensive instrumentation plan used in the depillaring operation for monitoring of strata behavior of mine for stability as well as roof bolts support behavior. The monitoring program was highlighted by successfully measuring the roof convergence during widening of gallery and pillar extraction for final coal excavation with 6.5 m working height. Roof displacement was also monitored after the installation of roof support.

Due to complicated geometry of some part developed panels and complex and slope procedures of pillar extraction (by diagonal slicing), rock mechanics and strata behaviour in bord and pillar depillaring workings are different from other common underground coal mining methods. Mainly two empirical approaches, CMRI Geomechanical Classification (CMRI-RMR) system and NGI Rock Mass Quality Classification (NGI-Q) system, are being used for design of support system for bord and pillar depillaring operation. CMRI-RMR system is used for design of support system in roadways during development stage of the mine and NGI-Q system is used for design of support during final extraction (depillaring). In the present study, a case study of NCPH Colliery R-6 Mine, SECL is discussed in details in respect of support design and geotechnical investigations during depillaring of panels. It has been aimed to collect relevant geotechnical information, to appraise the existing geo-mining condition and to estimate rock load at galleries, splits, slices and goaf edges in depillaring areas. An attempt has also been made to investigate the support resistance (SRF) value, which can be used to calculate the rock load for existing galleries or splits and slicing in depillaring, which is presently estimated using CMRI-RMR system.

The mechanics of load transfer in Bord-and-pillar mining is poorly understood, although the art of pillar extraction has evolved over the last 20 yrs through trial and error (empirical) or RMR, simple monitoring, and pillar- extraction experience in neighboring mines. Variations in geologic conditions and pillar- extraction methods and lack of comprehensive instrumentation programs for monitoring strata behavior during depillaring are among the factors that make it difficult to predict how strata will respond to pillar extraction.

Efficient and safe use of roof supports depends on the interaction between the designed support and coal strata. In particular interest to this study is the mechanics of strata deformation as influenced by geologic conditions, pillar extraction methods, and suitable support with load-bearing capabilities in strata control. To provide a better understanding of the mechanics of strata deformation, the as a investigator we have collected and reviewed measurements of convergence and stress in NCPH Colliery R-6 Mine of SECL for Coal extraction with Continuous Miner and have completed Numerical modeling for two typical pillar- extraction plans. The theoretical basis for designing support systems is often derived from a demand-capacity analysis whereby the capacity of the support system is extrapolated from field testing while the demand can be estimated using a variety of techniques ranging from simple gravity of wedges to sophisticated discrete element numerical modeling. As a result of the complexity and variability of rock mass such techniques relying on estimation of demand for the support system have important limitations. Rules of thumb and empirical approaches, despite their shortcomings, are still widely used because of their simplicity and robustness. Roof support's was calculated to show how install support contributed to the control of roof block movements. It was shown that overall stress and roof-floor convergence patterns were most influenced by the stiffness of coal-measure rocks and by pillar- extraction sequences and layouts. Support systems play a critical role in controlling the stability of both the immediate roof and the middle roof for a distance of up to 18 m above the seam. Install support provide sole strata control advantages over other types of secondary support by significantly reducing the time between mining and installation of secondary support. This study has been also helpful to exhibit role of the strata control for pillar extraction with blast free technology in thick seam mining.

3.1 Geological Description of the Field Site

Field Investigation was conducted under the boundary condition of NCPH Colliery R-6 underground mine is located in the Chirimiri Area of SECL in Korea district of the State of Chhattisgarh (Figure-3.1, shows location of field investigation mine site). It is bounded by latitude 23^0 11' 20" to 23^0 30' 20" N and longitude 82^o 19' 40" to 82^o 23 ' 08" E. as per Survey of India Topo Sheet no. 64 I / 8. The Project lies in the eastern part of Chirimiri coalfield at a distance of about 1 km from Anuppur-Chirimiri Railway line of South Eastern Central Railway. By rail it is 90 km from Anuppur on Bilaspur-Katni main line. An all weather road connects the project with important towns of Korea district. Bilaspur town, hosting the headquarters of SECL, is at a distance of about 200 km.

Figure-3.1: Location of Field Investigation mine site

After 2011-12, the estimated balance reserve is about 7.674 MT. The quality of the coal seams is B and C. With the present rated production capacity of 1.5 MTPA, the life of the project will be another 9 years, i.e., it will be closed by 2018-19.

3.1.1 Boundaries of the Mine: The following features bind the colliery area, (Figure-3.2, Shows key plan of field investigation mine site) i.e., the core zone.

North - West Chirimiri colliery and forest area

South - Kurasia colliery

East - North Chirimiri Colliery

West - Forest area

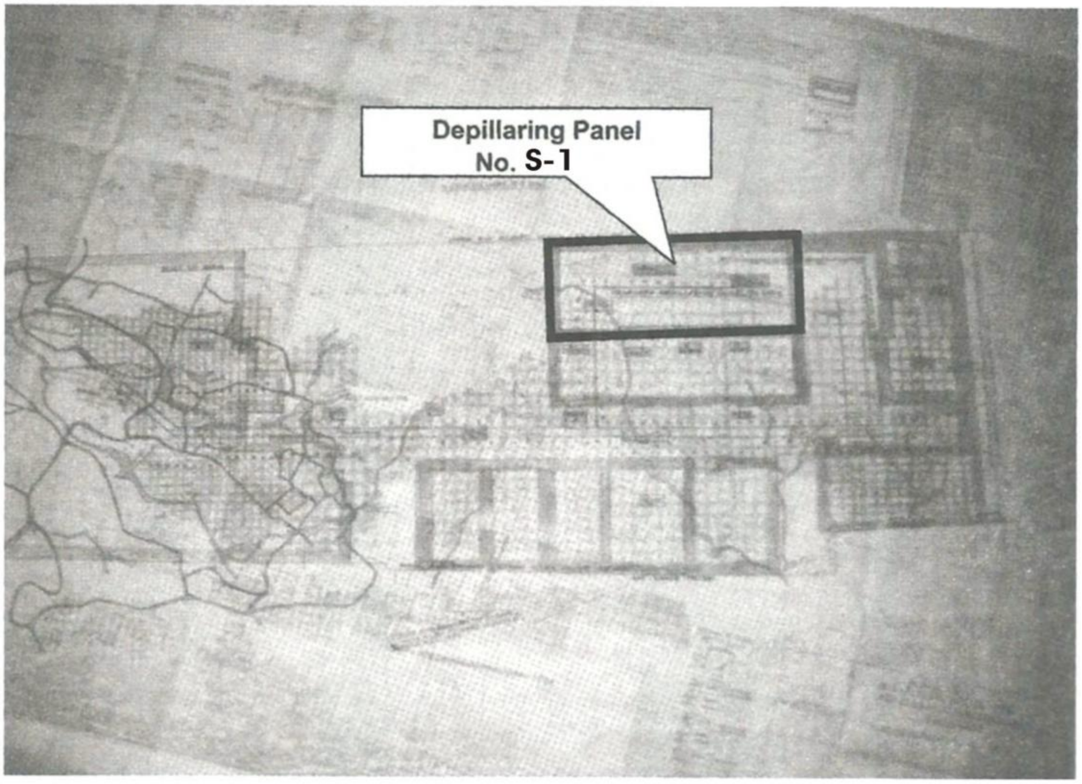


Figure-3.2: Key Plan of Field Investigation Mine Site

3.1.2 Topography and Drainage: The core and buffer zones of the colliery are marked by highly undulating topography with the highest RL being slightly more than 850 m. The colliery area is marked by the presence of a major hillock and its altitude is varying from 510 to 850m. The general slope of the ground is towards North-West. The area is drained by a number of first and second order drains originating from the hillock and then forming a nallah.This nallah (called Korea nallah) ultimately joins Hasdeo River near Manendragarh.

3.1.3 Geology: The litho-stratigraphic units encountered in the colliery leasehold belong to the Karharbaris/Barakars and the Deccan Traps. The Karharbaris/Barakars are represented by Sandstone, shale, carb shale and coal seams. The coal seams outcrop in the depressions and on the flanks of the well. They tend to follow the surface counter because of the rather flat dips. Study area is generally free from faults. An E-W trending fault divides the working into two parts. The area lying to the North and South of this faulted zone are known as R-6 workings and Amajharia & Haldibadi workings respectively. The fault zone narrows down towards East. The throw of the fault is about 30 m towards South in the areas adjacent to the North Chirimiri Colliery and increases towards East. The area forms a part of the shallow basin deposit with rolling dips of 10 to 40. Dip is almost flat in the central part of the R-6 workings. Figure-3.3, Shows part plan of investigation panel S-1, R-6 mine site.

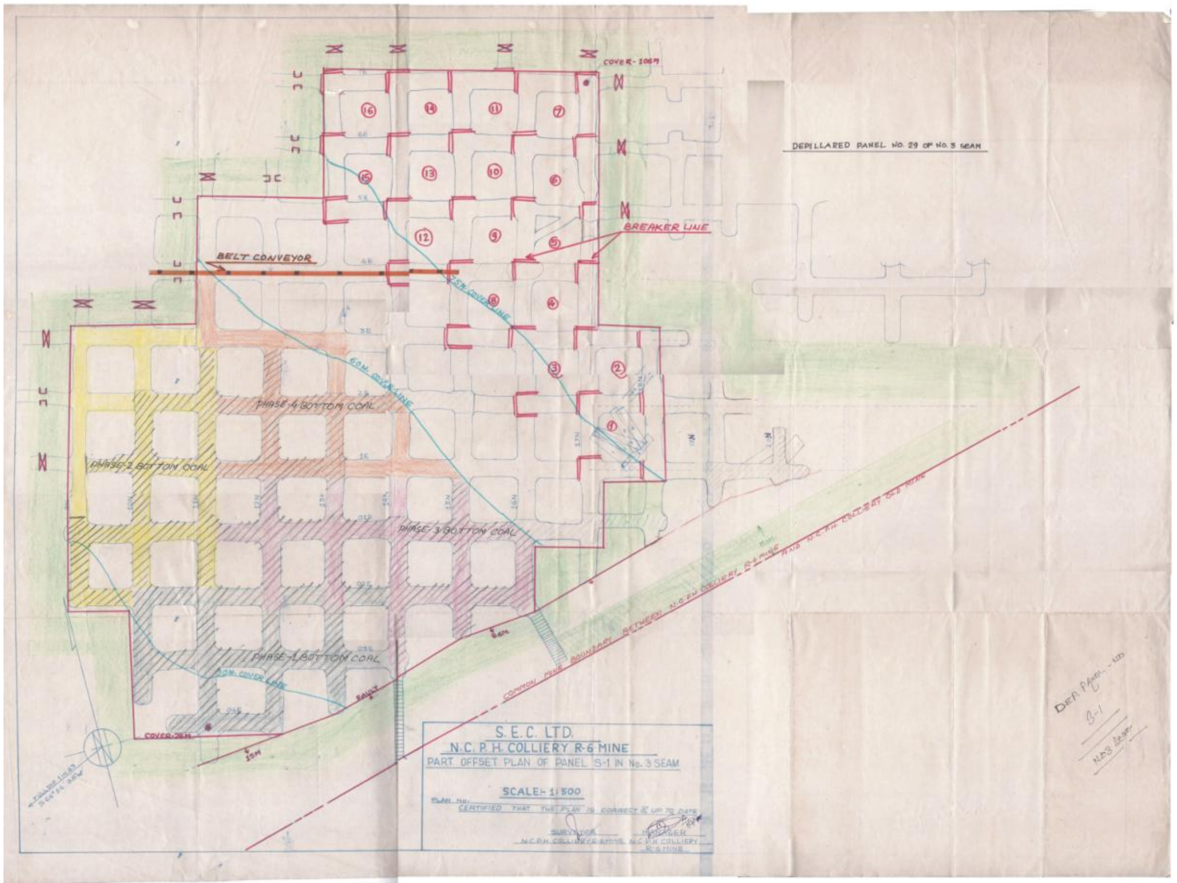


Figure-3.3: Part plan of investigation panel S-1, R-6 mine site

Dips up to 40 are observed in the area close to the North Chirimiri Colliery leasehold. The strike in this area is NE-SW and the dip towards NW. Figure-3.4, Shows geological details of seam position in N.C.P.H. Colliery.

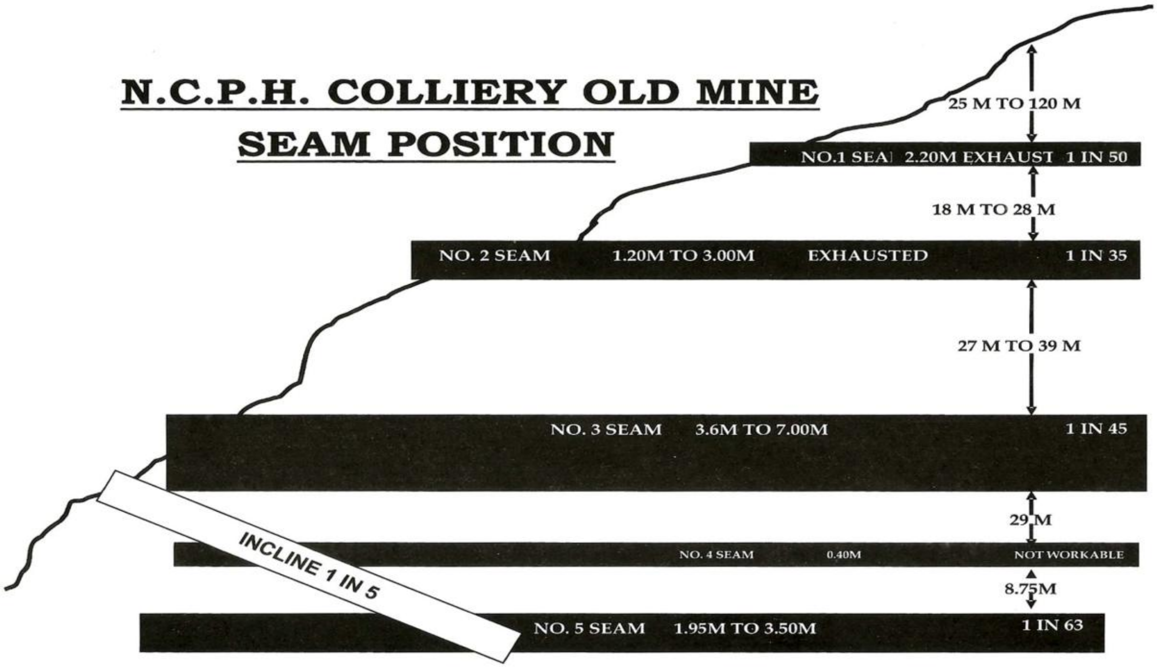


Figure-3.4: Geological Details of Seam position in N.C.P.H. Colliery

3.2 Geo-mining Parameters of Study Area

Roof lithology and structure plays an important role in contributing to unstable roof conditions. During mining, abrupt changes in the behavior of roof lithology and structure may give rise to a complex situation in which strata deformation and creep effects lead to further strata weakness. The knowledge gained by preparing of borehole core samples help in identifying areas prone to roof instability. The primary requisite to study a roof strata's lithology characteristics is to log the borehole core sample's geotechnical data, furnishing the details of core dips, laminations, fractures, the nature of contact between the litho-units, the average length of core, the percentage of Rock Quality Designation (RQD%) and structural features. The geological succession established in the mining block of the NCPH Colliery, R-6 Mine, and seam no.3 from the borehole exploration data is given in Figure- 3.5, along with the thickness range of each formation within the blocks as interpreted from the borehole's section.

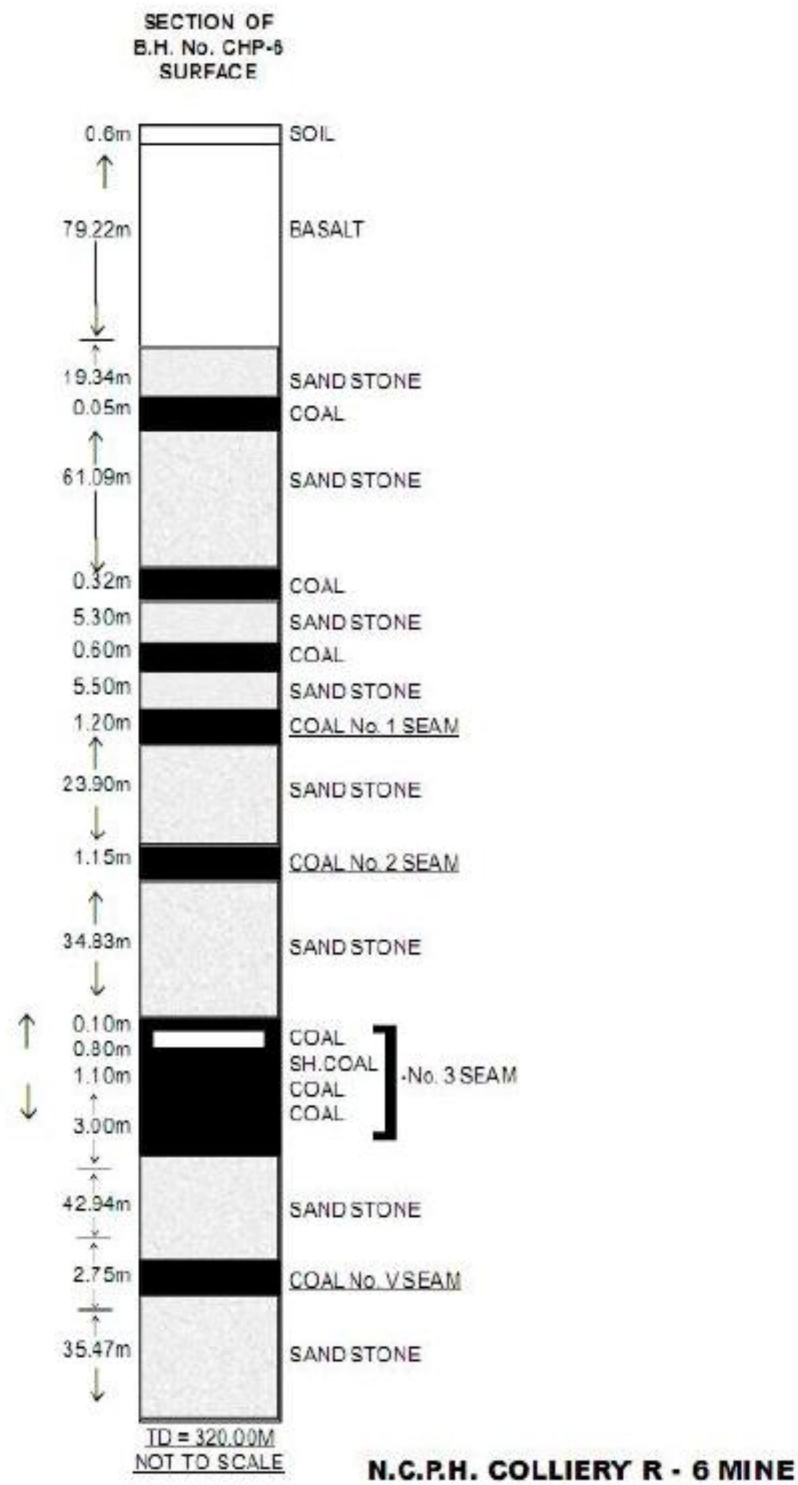


Figure-3.5: Intersected Geological Succession of NCPH Colliery, R-6 Mine and Seam No.3

3.2.1 Geo-mining details of N.C.P.H. R-6 mine in Continuous Miner Section

The following table shows specification under data collection and strata monitoring has taken:-

Table 3.1: Geo-mining details of the panel S-1 of NCPH Colliery, R-6 mine

Parameters	**Descriptions**		
Name of seam	No. 3seam		
Gradient of seam	1 in 40		
RMR	48.07		
Compressive strength	183 - 543 kg/cm^2		
Geological disturbances	Nil		
Method of working	Depillaring with caving by diagonal slicing with Continuous Miner		
Panel Dimension	156m level side x 96m dip side		
No. of pillars	24		
Nature of roof strata	7- 9m medium to fine grained sandstone		
Nature of floor strata	Fine to medium grained sandstone		
Development Details			
	Max.	Min.	Avg.
Seam thickness (m)	4.2	2.4	3.6
Depth cover (m)	106	60	86
Pillar size (centres),m	23.5x23.5	22x22	22x22
Gallery height (m)	3	2.4	2.5
Gallery width (m)	6.0	4.8	6.0
Depillaring Details			
Height of extraction	Full height of extraction		
	Max.	Min.	Avg.
Width of split (m)	Splitting of pillar not done		
Width of diagonal slice (m)	6.0	6.0	6.0
Width of rib pillar (m)	4.0	2.0	3.0

3.3 Mine Workings

NCPH colliery is situated in almost central part of Chirimiri coalfield. The colliery is working four coal seams in its two parts, which are separated by a major fault. In all the seams bord and pillar system with caving and partial extraction has been adopted. Brief details of the method of workings currently being adopted are given below.

3.3.1 Convention board and pillar system: In the convention board and pillar system the panels are being developed keeping in view the provisions in the Coal Mines Regulations 1957 for the size of the pillars and galleries and incubation period of about 9 months for depillaring life. During development the coal is being won by conventional solid blasting as per the directives from the Directorate General of Mines Safety (DGMS). The roof is being supported by 2.4-m long resin bolts. Depillaring of the pillars is being done by conventional stooking/slicing method with supports as per the systematic support system approved by the DGMS.

3.3.2 Continuous miner in board and pillar system: In the panels designed for extraction by the continuous miners the development had been done by these miners and finally the extraction of the pillars is also proposed to be done by these machines. Figure-3.6, Shows Bord and pillar with continuous miner working during development. During development and splitting the roof has been supported by 2.4-m and 1.8m long resin bolts.

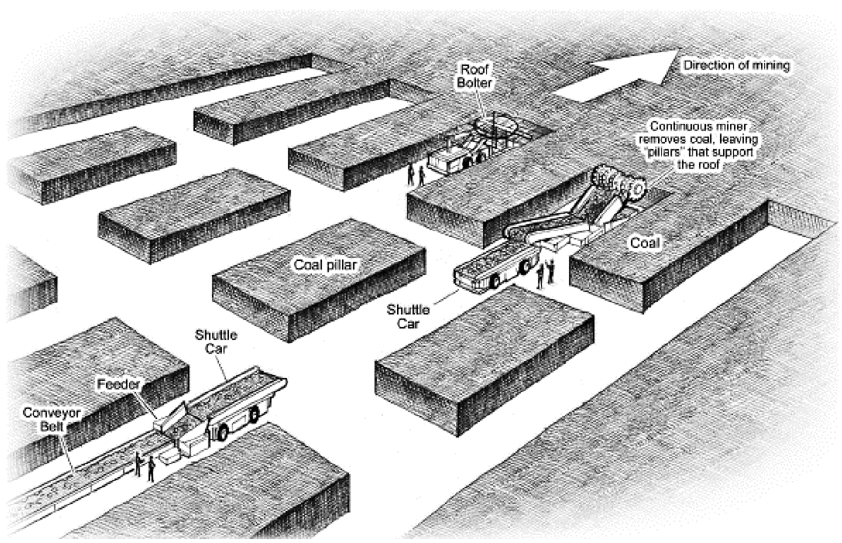


Figure-3.6: Bord and pillar with continuous miner working during development

A.) Development of the Coal Seam with Continuous Miners: In this mine R-6 there was usual practice of development of the seam along bottom of the Seam. But this panel S-1 had been developed along roof of the seam. So as seam was developed along roof of the seam with as height of 4 mtrs leaving coal at the floor. Some portion of coal seam already developed with the 4.8 mtrs gallery development of the Seam along bottom of the Seam with SDL thereafter introduction of continuous miner widening of gallery has been done for more space and its maneuverability for efficient utilization of the continuous miner. The following practices adopted in development of coal seam.

1. Size of the pillars should not be less than 22m x 22m centre to centre in Panel No. S-1 and solid pillar should not be than 16m x 16m corner to corner. The width of galleries should not exceed 6m and height should not exceed 4.0m while leaving coal along the floor.
2. Development should be done always along the roof and if possible some coal may be left on the floor.
3. The maximum cut out distance should not exceed 12.0 m.
   (Cut out distance – Continuous Miner cut maximum 12 mtrs from the last line of support)
4. Support system – There was four rows of non-retractable roof bolts of 1.8 mtrs in length and bolt dia 22mm in dia, the distance between the two adjacent rows of roof bolts should not be more than 1.5m and that between two adjacent bolts in a row should not be more than 1.2m.

LAYOUT FOR CUTTING SEQUENCE

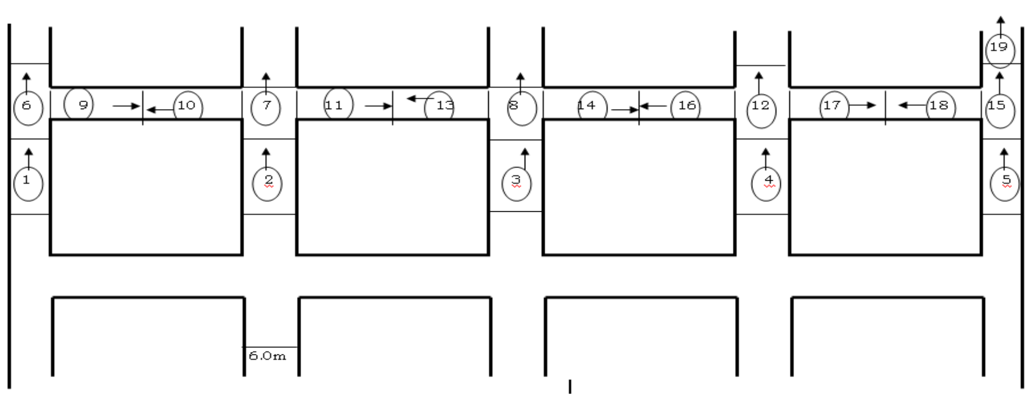


Figure-3.7: Cutting sequence continuous miner working during development

5. Formation of pillars should be commenced from left to right in the panel.
6. The Continuous Miner should cut the left gallery as shown as (1) in sketch. After cutting 12 mtrs it should be trammed to position as shown as (2) in the sketch.

7. While continuous miner cut the position at (2), quadbolter supported the face already cut by the continuous miner shown as (1).
8. After cutting position no. (2), the continuous miner trammed to position no. (3), and the quadbolter supported the position no. (2).
9. Likewise Continuous Miner cuts and the quadbolter supports and follows the sequence of cutting & supporting as shown in the layout plan that all the headings were advanced simultaneously.
10. One cut of the Continuous Miner was of 12 mtrs in length.

B.) Depillaring with Continuous Miner: After successful development of the seam it was considered for extraction of the pillar. There were two alternatives for depillaring the panel S-1. The two alternatives were –

I. Splitting and Fender method.
II. Diagonal Slicing.

II. Diagonal Slicing

In this method cutting the pillar was done in diagonal fashion from the original gallery without splitting.

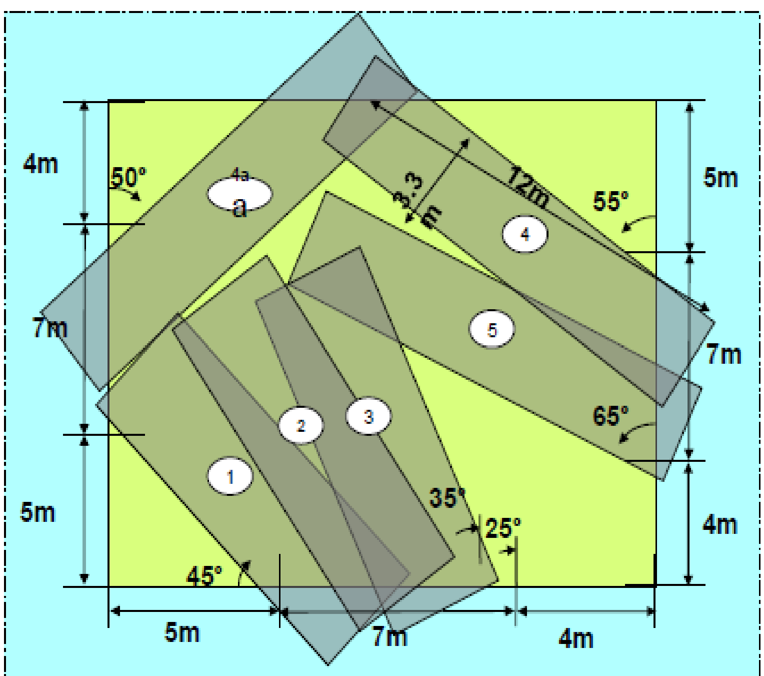


Figure- 3.8: Pillar Extraction-Slicing Sequence with dimensions

In diagonal slicing method, cutting the pillar was done in a diagonal fashion. It was to ensure that the Continuous Miner did not go too deep into the pillar. It required that a cut should be taken from the side or back of the pillar some time before it was finally extracted. The cuts were of 12 meters in length. In this method it was possible to extract these pillars without splitting the pillar. As this method did not require split that needed supporting. It speeded the rate of extraction and was significantly safer as less time was spent in the pillar. The split method had an additional risk as this required to place the support personnel within the split.

So for the above reasons it was recommended that diagonal slicing method should be adopted for pillar extraction in S1 Panel.

Extraction Method of coal by diagonal slicing: In the S-1 panel straight line of extraction had been adopted. In the first step a diagonal slice of 3.3 meters width and 12 meters of cutout distance were commenced from the original gallery. The angle was about 45^o to 50^o from the original gallery. It is shown in the below picture. After taking diagonal slice, bottom coal of the diagonal slice was taken out. Height of extraction should not exceed 6.5 meters. This was done by Continuous Miner by making a gradient from the original gallery to the diagonal slice.

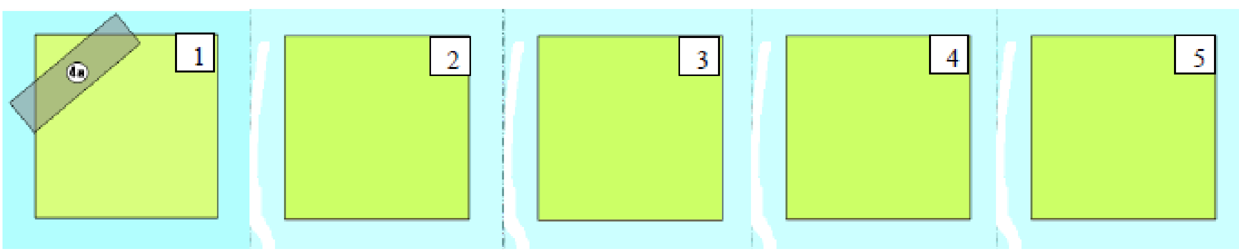


Figure- 3.8.1: Step-I, Pillar Extraction-Slicing Sequence with dimensions

After completion of no. 4a diagonal slice (as shown in the picture below) from the original gallery, shifting of the continuous miner (as shown in the picture below) was done and the diagonal slices of no. 1, no.2 & no.3 was driven. The cutout distance was 12 meters. The angle was 40^o to 50^o from the original gallery of the diagonal slicing. After driving slice no.1, no.2 & no.3 bottom coal was extracted. Height of extraction did not exceed 6.5 meters.

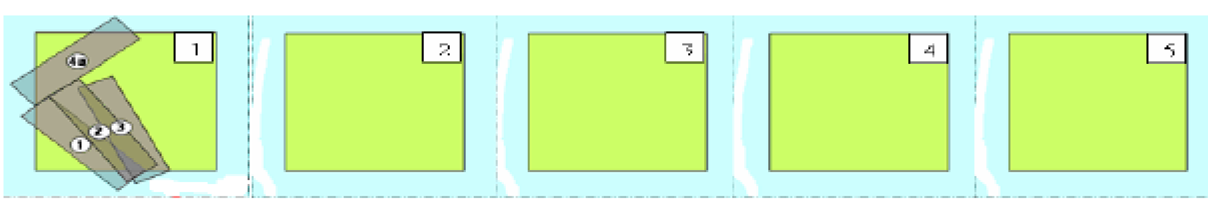


Figure- 3.8.2: Step-II, Pillar Extraction-Slicing Sequence with dimensions

After completion of no. 1, no.2 & no. 3 slice of pillar no. 1 a diagonal slice of no. 4a of next pillar (as shown in the picture given below) was driven. The cutout distance was 12 meters. The angle was 45^{o} to 50^{o} from the original gallery of the diagonal slicing. After driving slice 4a of next pillar, bottom coal of the diagonal slice was taken out. Height of extraction was not more than 6.5 mtrs.

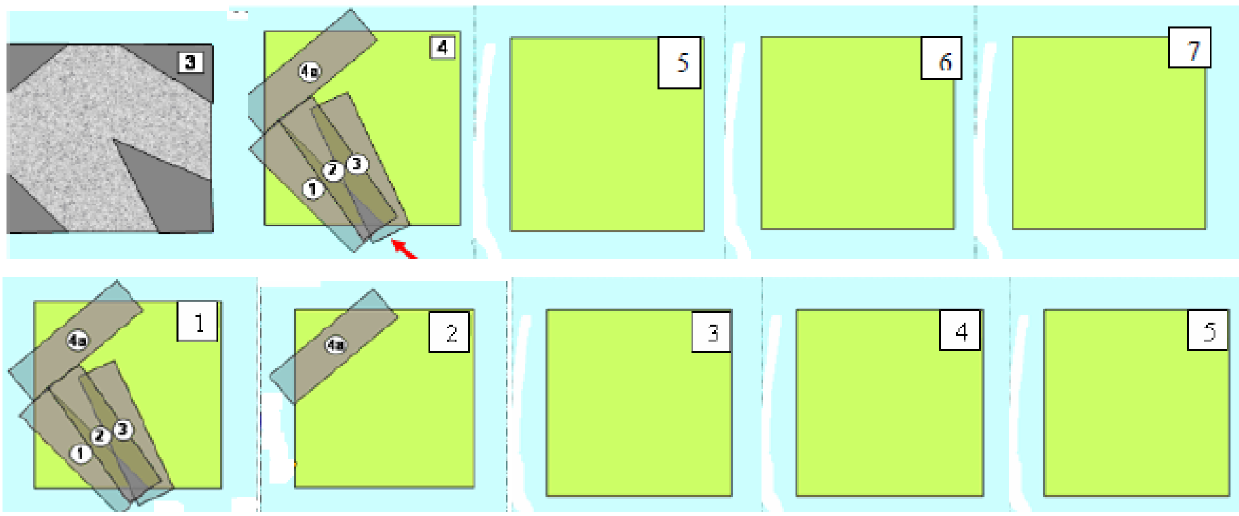


Figure- 3.8.3: Step-III, Pillar Extraction-Slicing Sequence with dimensions

After completion of no. 4a diagonal slice of the next pillar, the diagonal slices of no. 4 & no. 5 of the first pillar were driven. The angle was 55^{o} to 65^{o} from the original gallery of the diagonal slicing. After driving slice no. 4 & no. 5 bottom coal of the diagonal slicing was taken out. Height of extraction was not more than 6.5 meters.

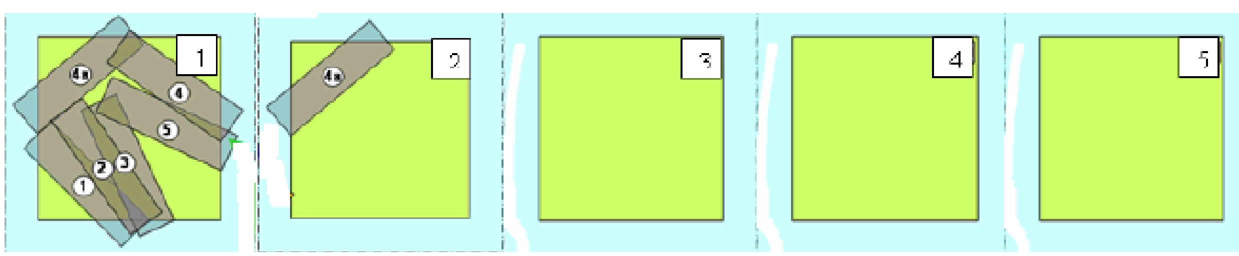


Figure- 3.8.4: Step-IV, Pillar Extraction-Slicing Sequence with dimensions

In this fashion extraction of the coal of pillar no. 1 was done and the same above sequence for driving the slices no.1, no.2 & no.3 of the next pillar was repeated. Driving the slice no. 4, no.5 & no. 4a of the above said sequence was also followed.

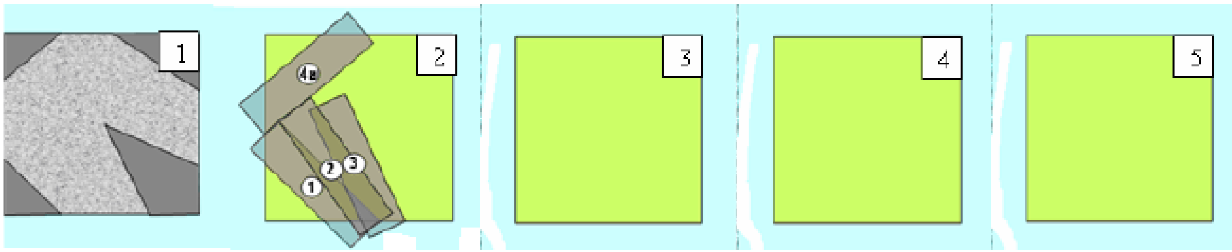


Figure- 3.8.5: Step-V, Pillar Extraction-Slicing Sequence with dimensions

The speed of the rate of extraction by driving the diagonal slicing was maintained. The width of the slice was not more than 3.3 meters at a time. There was no bolting during the extraction of pillar. Before commencing of extraction of pillar, only breaker lines of roof bolts was erected. Roof bolt breaker lines were consisting of two rows (6 bolts in a row) of rock bolts having 2.4 meter long, 22 mm diameter, and full column resin. The spacing between two bolts in row and between rows was 1 mtr.

3.4 Support System of Study Area

In our study area two types of supporting system in S-1 panel earlier system during development has taken quick setting cement capsules with roof bolting in SDL deployment then after mine management has taken decision to deploy the continuous miner rehabilitation from the Anjan hill mines of Chirimiri Area itself. Figure-3.3 shows S-1 panel layout for working of continuous miner with diagonal slicing coal pillar extraction.

3.4.1 Support System in Earlier developed S-1 panel

Supporting with quick setting cement capsule roof bolting is acceptable in this panel. The continuous miner (CM) has cut for a maximum distance 12 m beyond last line of support by radio operated remote control or umbilical cord. Then the CM has trammed to another adjacent heading. After 12 m of cut the freshly exposed roof has been supported by roof bolts with quick setting cement capsules. All working faces have been supported by four roof bolts in a row of 1.50m length and 20mm dia. The distance between the two adjacent rows of roof bolts any place not is more than 1.50M and the distance between two adjacent bolts in a row not is more than 1.50M. Support of junctions immediate out bye of the development face: - 25% extra. All faults visible slips and cracks in the roof: - It has been supported with roof bolts at an interval of 1.5M between rows of bolts as well as between two consecutive bolts in a row with ‘w’ straps of

3.15mm thick and 2.40M in length. The roof bolts has been 20mm in diameter and not less than 1.50M in length. The difference of diameter between the hole and the roof bolts not is less than 3mm and not more than 6mm. Each hole had been set of quick setting cement capsules.

3.4.2 Support System in widening of gallery and depillaring in S-1 panel

The standard roof bolt used throughout the R-6 Mine was 1.8m for original gallery and 1.5m for split gallery, and 22mm diameter rebar with a 150mm thread and a range of 14T-22T in yield strength of the rod. The bolt was installed in a 25mm diameter hole with quick setting cement capsules for nominally full encapsulation. A crimped nut and antifriction washer plate was used. Successful installation of the bolt requires the drill operator wait a specified period for the quick setting cement to set before completing the installation by tightening the nut. Figure- 3.9, 3.10, 3.11, & 3.12: Shows the systematic support rules during development and depillaring.

A study of typical installations revealed the following problems:

- The hole was often drilled too long, preventing full encapsulation.
- The quick setting cement was not inserted to the top of the hole.
- Mixing was not continued for the specified time.
- The hold period was too short.
- Torturing often resulted in the bolt being pulled down the hole, permanently reducing the bond strength.

It was observed that correct installation of the system was not taking place. The bolt system used was difficult to install correctly and prone to operator error. The resulting installation was likely to have very low bond strength. In-mine roof bolt systems did not incorporate such features as effective torque nuts and deformable end plates, which provide quality control during installation and information on the development of end loading.

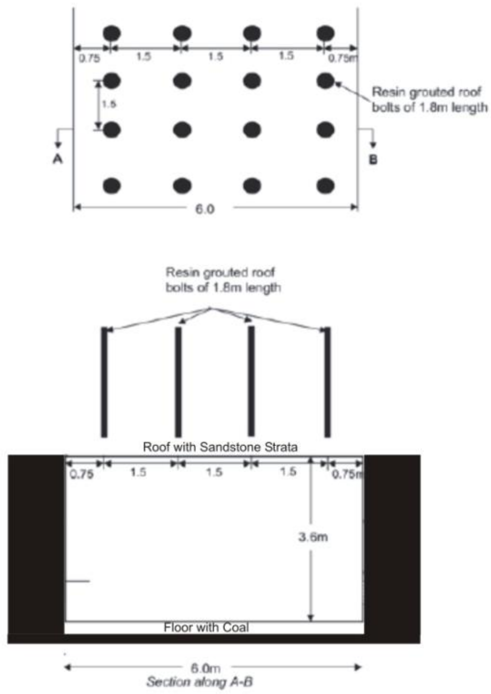


Figure-3.9: Support system in development stage after widening of galleries

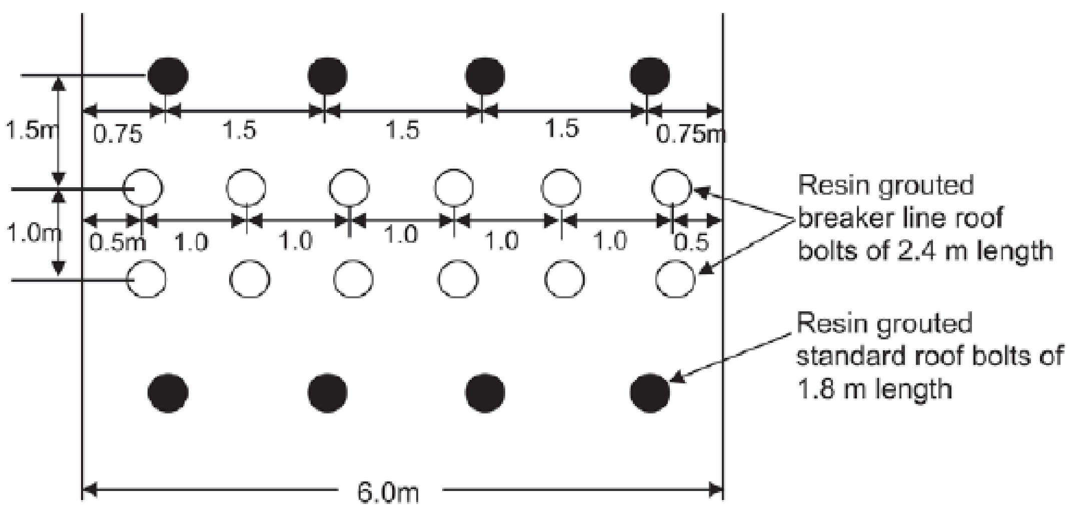


Figure-3.10: Breaker line support system in depillaring galleries

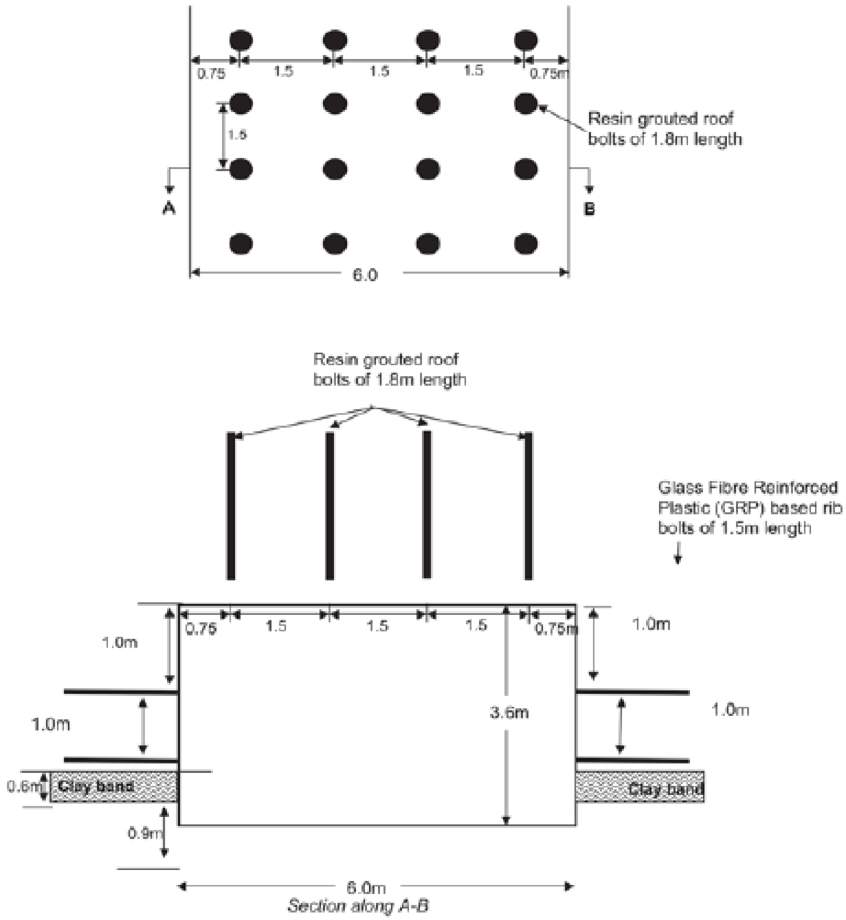


Figure-3.11: Support system in original galleries before commencement of depillaring

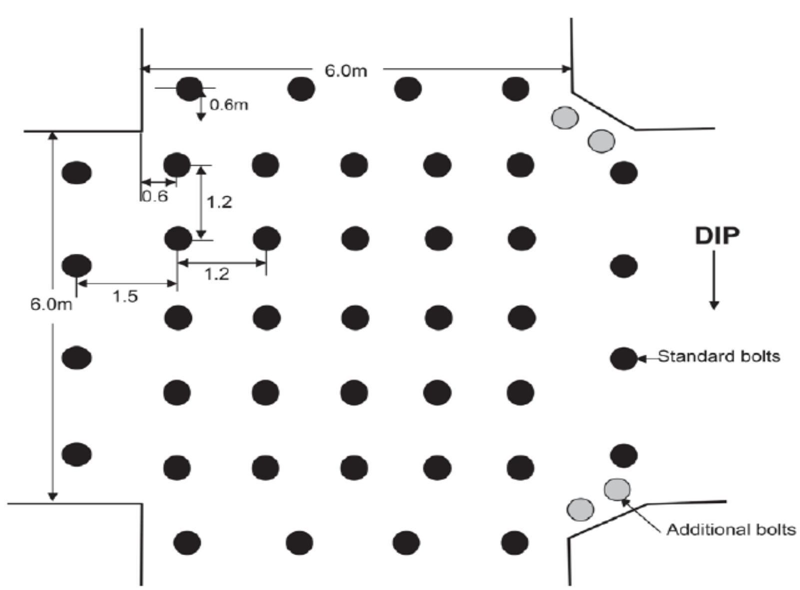


Figure-3.12: Support system at Junctions

3.5 Monitoring of Strata Behavior

This field investigation-analytical program was undertaken to determine the fundamental strata behavior during room and pillar mining of coal. NCPH Colliery R-6 Mine of coal company South Eastern Coalfields Limited was selected for the field investigation to analysis of strata behavior with respect to convergence for verifying existing support system. The coal seam was approximately 86 m underground and was located in a region of rolling hills. Seam thickness varied from 2.4 to 6.5 m. mining was done with of using a continuous miner. The Mine sites panel S-1were instrumented; two sites with good roofs and one with a weak roof. For the two sites with good roof, one junction in a development area, and two junctions in a production area were instrumented. One junction in the panel S-1 site with poor roof conditions was instrumented. Three kinds of measurements were made in each intersection; roof bolt loads, differential displacement of roof strata (sag), and roof-to-floor convergence. This instrumentation was placed as the working face advanced. Details of the boundary conditions of the panel and field observations at different location in the panel during its depillaring are mentioned in this part as an example of Combined Instruments Approach (CIA). Suitable designed arrangement with the field data was obtained. The analysis included excavation effects also. A clearer understanding of the load transfer and ground movement phenomena was obtained from the studies. Roof bolt loads and sag showed excellent correlation with excavation and exhibited no time dependent effects. Roof-to-floor convergence exhibited time dependence and the fastest convergence rate occurred immediately after excavation and became constant after several weeks. Changes in roof bolt loads during excavation were strongly influenced by movement of the anchor point.

For the most part, excavation design at this mine is dictated by the size of the mining equipment. At the NCPH Colliery, R-6 Mine the roadway excavations were typically 6m wide; this was the optimal size for the equipment used (3m wide continuous mining machines). This size is typical for coal mines using similar types of mining equipment. It is possible to create smaller excavations; however, this is not as cost efficient, as the full capacity of the continuous miner would not be used. As well, coal transportation is slower on account of reduced maneuverability of the equipment in the narrower roadways.

Experience at the R-6 Mine showed that 6m wide excavations in sandstone, at depths of cover less than 100m, were generally stable once they were supported. Excavations that were stable when they were first developed usually remained stable. Most unstable excavations were unstable as soon as they were developed. The main exceptions to these observations were areas that deteriorated by weathering, and areas with adverse geologic structure.

Empirical formulation of support system, based on in situ measurements, is an acceptable way to estimate rock mass behaviour and roof bolt support system. This study has made for assessing convergence in existing support system during development as well as extraction of coal pillar by continuous miner and analyzing the convergence of roof strata due to thick seam (6.5m) coal extraction under specified geo-mining conditions for ensuring safety and productivity. This study was also verified with simulated numerical models for suitable support in the existing mining operation. Vibrating wire stress meters were used at strategically place for underground monitoring of the stress development with increase in dimension of excavation for reference. The quality of the overlying roof strata is assessed through geo-technical logging and testing of physico-mechanical properties of the procured core samples in Company laboratory. Attempts were also made for in situ testing of the rock mass by borehole rock indentor. In a stable environment the rate of convergence is small and usually remains steady or decreases with time. Whereas this shows increasing trends with increase in dimension of excavation mainly depends on overlying strata characteristics. Generally, it maximum value was observed just before the major roof fall in the panel. A simple empirical model has been studies regarding support system and it effectiveness during extraction, because there was no additional support installed during extraction of pillar, only commencing of extraction of pillar; roof bolt breaker lines was erected. It was observed that support of roof bolts installed during development have got considerable influencer with final extraction of pillar with diagonal slicing method. Impact of thick seam mining with existing support is observed to be the most serious factor for the safety of working under a hard and massive roof rock mass. The strata monitoring program included three main components:

iv) Monitoring of convergence during widening of gallery 4.5 m to 6.0 m.
v) Monitoring of convergence during extraction of pillar by diagonal slicing.
vi) Measurement of load on support for verifying stability of strata.

3.5.1 Monitoring of convergence during widening of gallery 4.5 m to 6.0 m.

For deployment of continuous miner under in panel S-1 at NCPH colliery R-6 mine gallery was widened from 4.5 m to 6 m figure 3.4 shows the part plan of gallery widening along with support strategy of widen (6m) gallery also illustrated. As per visual observation, the roof in same areas of this mine has damaged, in the form of fracturing, appears to be random in nature. In the panel S-1 selected for the gallery widening for deployment of continuous miner, there was no roof fracturing or any other obvious evidence of light horizontal stress.

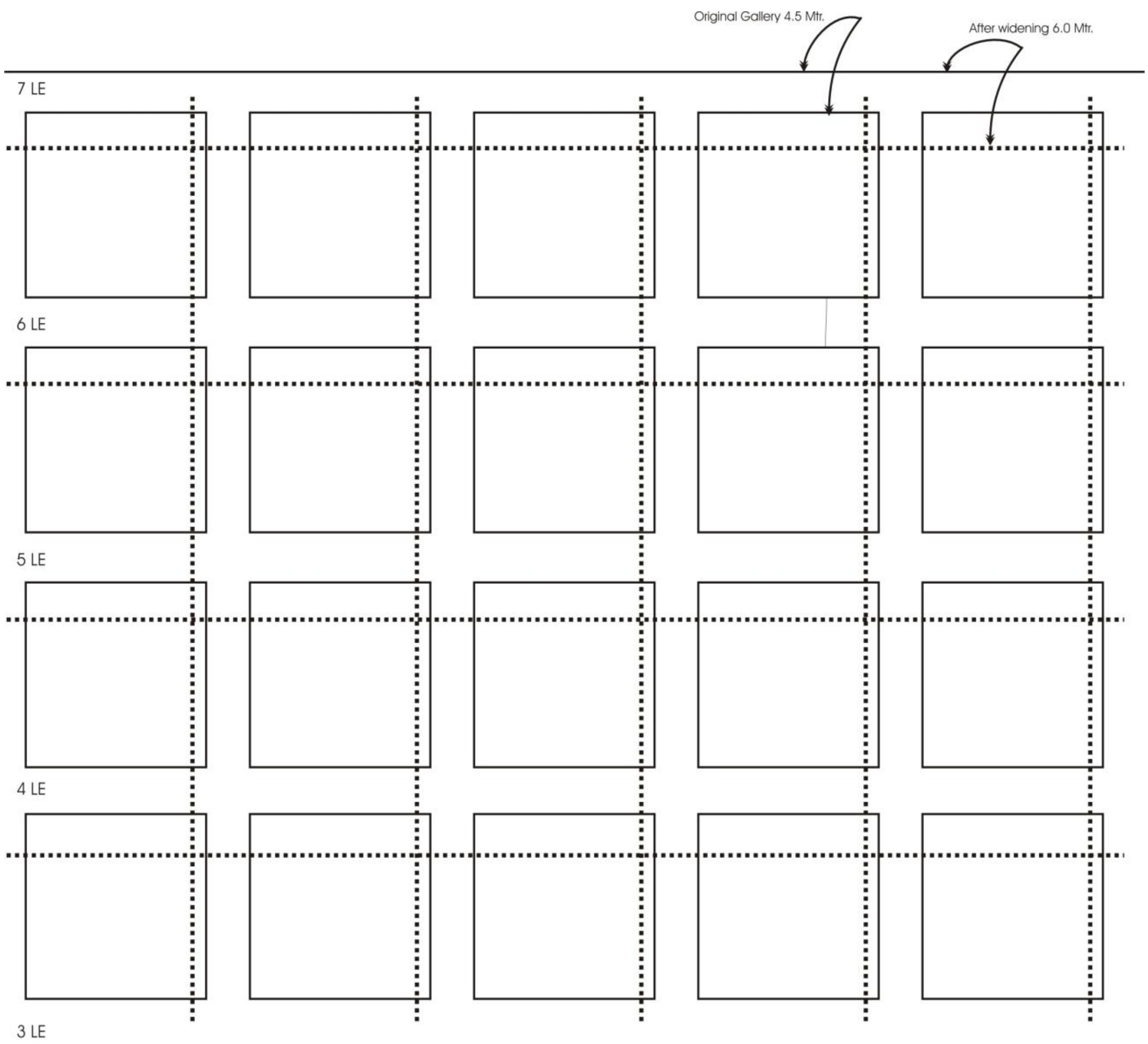


Figure-3.13: Shows the part plan of gallery widening

The immediate roof was thick competent sandstone unlike the roof within a couple of meters from the investigation site. Bearing in mind that is order for failure to take place the stress acting on a material should be greater than the strength of it. Therefore although the stress was probably (not measured) some as anywhere else on the mine, because of this competent sand stone the stress damage may have not been seen in the area.

The investigation site was an existing board used as a working place. The board and adjoining gallery were carefully examined. No significant geological features were observed that could adversely affect the roof stability in the immediate area. The roof was supported using 22 mm damage 1.8 m long full column resin both four bolts in a row with the row 1.2 m apart. The mining operation was carried out by drill and blast for widening of gallery.

3.5.2 Instrumentation plan during widening of gallery

The original board was 4.5 as wide and after in 6.0 in widen. Two sets of instrumentation were installed in the roof approximately 2.0 m from junction point of widen face. Each set consisted of a 7.3m deep sonic probe extensometer and two tell-tales were also installed at each position to monitor the strata between the roof layer along with 1.8 m and 2.7 m elevations.

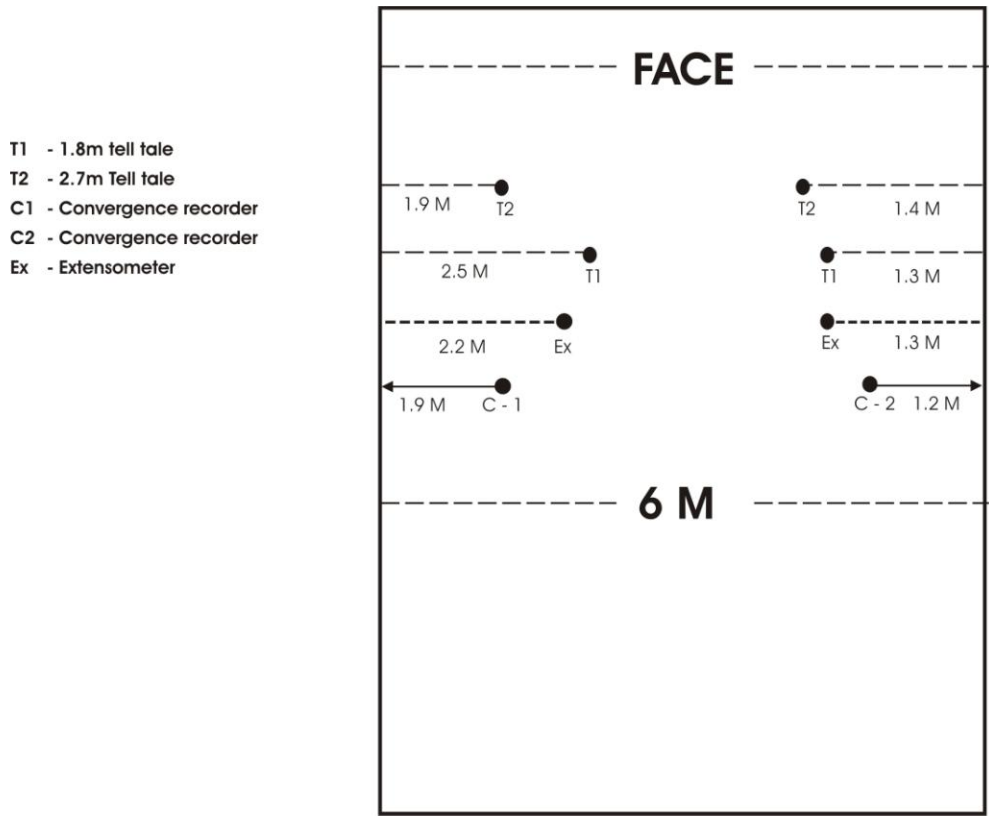


Figure- 3.14: Instrumentation layouts during widening of gallery

The instrumentation layout is shown in figure-3.14. Two sets of convergence recorder also set each side of gallery with 1.9 m and 1.2 m from side wall. The roof and side wall conditions could only be assessed during the widening operation and an analysis of strata behavior during widening of board width. The purpose of the sonic probe extensometer was to gather detailed information of the roof behavior as the board width was advanced and the widen gallery formed. It was also anticipated that some additional readings would be taken as the gallery widening commenced and for as long as it was safest to enter the area, if roof bolt support was installed. When the face was advanced, very small displacements were recorded close to the roof skin at both the locations. The total value at the side hole was 1.0 mm while 2.0 mm was recorded at the centre line of instrumentation. This fixed convergence points were installed primarily to be able to continue to monitor the roof remotely, during and after the gallery widening operation. Four tell-tales were also installed to monitor the same section of roof as the fixed observation points. A period after the initial installation very few data had been collected due to operational difficulties in observation of site performance of instruments. However summary data has been compiled table no. 3.2 (see the widening data).

Table no. - 3.2: During Widening of gallery width 4.5m to 6.00m

(Comparision of convergence station at 0.5 m & 2.5m Anchor)

STATION	NO.OF DAYS OPERATED	CUM.CON
C1	13	4
C1A	13	3
C2	26	6
C2A	26	6
C3	32	7
C3A	32	7
C4	35	9
C4A	35	8
C5	35	12
C5A	35	10

3.6 Monitoring of convergence during extraction of pillar by diagonal slicing

There is no "typical" underground mining condition during depillaring, which can be used as a benchmark to decide the instrumentation scheme for the strata control investigation. However, from stability point of view, underground coal mining generates mainly three types of structures. These three structures may be termed as: Long, Medium and short term stable structures (Singh et al., 2010b). Underground structures like pillars and galleries due to primary developments come under the first category while the applied/erected support fall into the second category and the structures like rib and slice belong to the third category. The sophistication and remoteness of the monitoring instruments increase with decrease in stability of the structure. Instruments placed to monitor performance of structures of the first two categories need not to be of remote type as the area around them remains mostly accessible and safe. Further the required time interval between two consecutive observations of an instrument in and around a stable structure need not to be very small and, generally, shift wise readings serve the purpose. Here, simple manual/mechanical type of instruments can provide the required information. However, the monitoring of short-term stable structures demands remote type instruments, mainly due to the hazardous nature of the rock failure in and around these structures. Also the frequency of observations is to be very high in and around the short-term stable structures like rib and slices. Here, it is preferred to be continuous in time due to the pace of the associated strata equilibrium dynamics around the goaf edge.

Success of a stratum monitoring scheme depends on large number of factors and even, sometimes, support of an underground miner becomes vital for the life of an installed instrument. Performance of instruments, applied for a strata control study, is dependent upon geo-mining conditions of the site (Singh, *et al.*, 1996; Singh and Singh, 1999b) in addition to the percentage of extraction in and around the instrumented site. Geological, technical and operational factors bring a number of threats for the safety of the applied instrument for an underground investigation. However, on the basis of experience of strata control monitoring at different coalfields of the country (Sheorey, et al., 1995; Singh and Singh, 1999a; RMT Report, 2003), following factors need special attention at the planning and instrument selection stage for an underground investigation programme:

i. Underground environment,
ii. Dimension of excavation,
iii. Depth of cover,
iv. Characteristic of roof rock mass.

Out of these four factors, the first two are common to all underground workings but the last two are bit site specific. Most of the depillaring operations in India are going to be under competent roof strata and variation in depth of cover is likely to be from 100 to 300m. It is a simple fact that the stability of excavations close to surface is mainly controlled by geological structures while the stability of deeper excavations is more influenced by the properties of intact rock and pre-existing stresses. Depth of cover has significant impact over the in situ stress condition, depositional compactness and geo-physical properties of rocks. Therefore, the depth of cover affects response of underground structures during strata equilibrium dynamics in and around a depillaring face, which ultimately, influences the nature of the strata monitoring instruments. The behaviour of overlying roof rock mass influences the characteristics of associated mining structures of an underground mine.

3.6.1 Instrument Details during extraction of pillar

Strata control investigations in NCPH Colliery R-6 Mine, mainly, involves following instruments: (a) Load cell, (b) Convergence indicator (c) Rotary Tell Tale/Dual height tall tale (d) Stress meter (e) Strain gauge bolt and (f) Magna sonic Extensometers/rib extensometer. All these instruments are used to gather, mainly, two types of information in and around the excavation: (a) stress/load and (b) strain/deformation. Different electro-magnetic sensors along with some site-specific mechanical arrangements are installed at different prefixed positions in a depillaring panel before the commencement of pillar extraction. The positions of these instruments are selected in such a way that they remain undisturbed during extraction of the pillars. Sometimes special arrangements are made to protect these instruments even inside the goaf till roof fall. Altogether following four types of sensors are used for monitoring strata behavior with instruments:

a) Linear variable differential transformer (LVDT),

b) Strain gauge,

c) Rheostat/variable resistance,

d) Vibrating wire.

Out of these four types of sensors, the first three types of sensors provide analogue signal while the last one sends only frequency (Singh et al., 2010b), which is a fixed value like digital information. It is very difficult to carry the analogue signals of the first three types of sensors to a distant safe place without noise and distortion. However, the frequency obtained by the vibrating wire sensor does not get affected during this transmission. This feature of the vibrating wire sensors makes it superior to other sensors because it is always desirable to monitor instrument’s reading from a distant and safe place during depillaring. The scope of instrumentation and monitoring utilizing a data logger interfaced with a computer is enormous. However, due to different technical and financial reasons, most of the Indian depillaring faces adopt manual monitoring approach. The connecting cables of the installed instruments are taken out of the working to a safe place and frequency of manual observations is increased after encountering a large overhang of roof inside the goaf. Conventionally, a number of instruments are installed at different selected stations inside the panel and their readings are manually monitored with face advance. Manual monitoring provides discrete readings, which may not be suitable to project the likely behavior of overlying strata. On the basis of different field experiences, a simple process called Combined Instruments Approach (CIA) is developed for better analysis of the manually observed data of underground instrumentation.

3.6.2 Mine Plan and Part Plan of Instrumentation in Panel S-1

Convergence indicators and, load cells were installed in 1LE, 2 LE, 3 LE, 4 LE, 5 LE, 6 LE and 7 LE in the CM panel S-1 were installed at an interval of 10 m along the levels. In addition to the above, instrumentation for observation of bed separation was done at 2 LE, 3 LE, 4 LE, 5 LE etc, All necessary records are being maintained from starting of extraction of the panel. The data is being analyzed regularly and necessary steps are being taken for better strata control. The observations are carried out manually every day. Location of strata behavior monitoring stations commissioned in the panel is shown in Figure-3.15

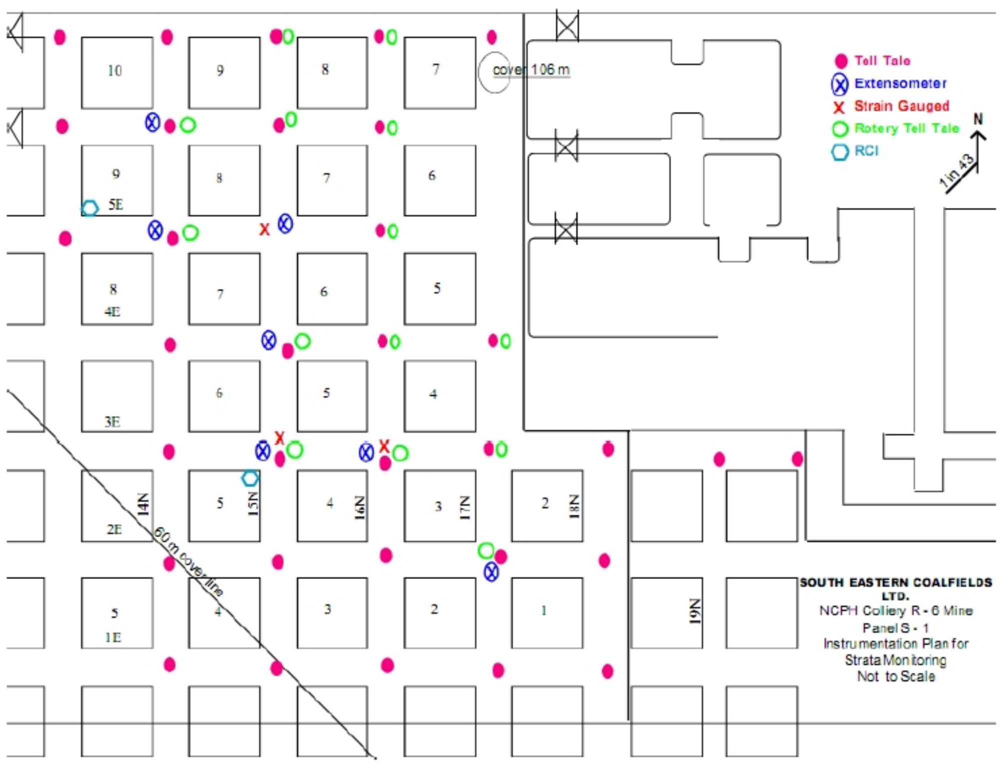


Figure-3.15: Mines Plan and Part Plan of Instrumentation in Panel S-1

Stress meters (vibrating wire type) were installed in a horizontal hole drilled (Figure-3.14) across each selected observation pillar. The position of a stress meter inside the pillar was chosen in such a way that they remain in the center of the stooks/ribs even after diagonal slicing/stoking of the pillars.

Figure-3.16: Installation of a vibrating wire stress meter in a horizontally drilled hole.

3.7 Measurement of load on support for verifying stability of strata

Numbers of load cell was placed during widening of gallery as well as extraction of pillar by diagonal slicing on a rock bolt to study the load history. The bolt was installed with the load cell at the time of after installation roof bolt during widening of gallery and in the time of pillar extraction approximately 40m back from an advancing goaf. The load cell was left in place until just before it passed into the goaf, at which time it was recovered. The load history is shown in Appendix- II. Bolt tension and load were found to remain essentially unchanged throughout during widening of gallery and also showing stable excavations. Whereas during extraction of coal in a thick seam (6.5 m) by continuous miner with diagonal slicing some measurement shows roof convergence and as well load on support which is not significance to safe excavation stability. This finding is analysing in next chapter 4 for consistent with the strata and support behavior with respect to convergence study. As can be seen, the load on the rock bolt did not change from the time of installation to the time of removal (just prior to passing into the goaf). Since there was no increase in load on the bolt, it can be concluded that there was no deformation in the roof rock even as the goaf approached. This also infers that there was no change in the induced stresses in the immediate roof.

3.8 Collection of Data at the Field Site

All the instruments lying within two pillars from the pillar under extraction had recorded daily except for remote convergence indicator& stress meter which should be recorded on shift basis. Recording of the instruments had done on alternate day basis. A brief summary of the instruments' readings in the S-1 panel of the NCPH Colliery R-6 Mine are also enclosed in Appendix- I roof convergence measurement data during widening as well as extraction of pillar by diagonal slicing and Appendix- II load on roof bolts support for further references.

3.9 Observations of strata behavior

During widening of gallery from 4.5m to 6m, the maximum deformation observed was 2mm at an advancement of 8m from the face. At the junction, the maximum deformation was found to be 6mm where local faults exist. During depillaring operation, the maximum and minimum convergence over a rib pillar, 16m span is observed to be 4mm and 1mm respectively. The maximum and minimum strain developed in the pillar sides are 3mm and 1mm at a span of 6m and 19m respectively. The maximum deformation of strata during depillaring is observed to be 9mm. At a depth cover of 75m, the maximum and minimum stresses on the 22m x 22m pillars are observed to be 3Mpa and 1.6Mpa respectively. In figure-3.17, Shows the graph of the Convergence vs No. of days during widening of galleries, where Y- Convergence in mm and X- No. of days during widening of galleries.

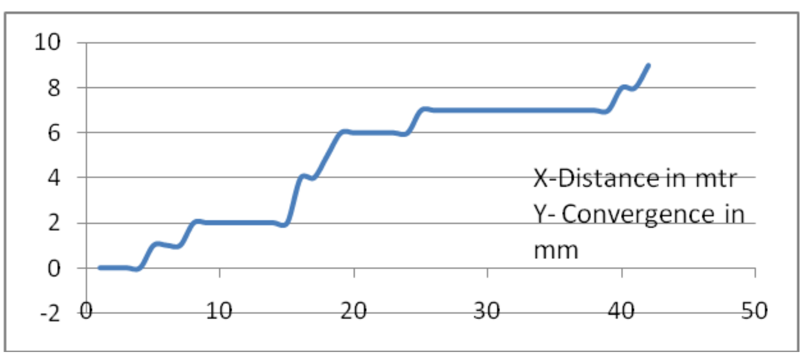


Figure-3.17: Covergence / Distance in meter

During the depillaring operation being carried out in NCPH R-6 mine, the observed trend of deformation near the rib pillars shows that there is no local fall till an advancement of 30mtr having a convergence of 16mm showing a minor disturbance in the roof strata but as half of the panel is extracted, the deformation has increased to 19mm resulting in the first fall in the panel. This clearly indicates the increase in stress in the rib pillar after advancing of 140m. The strain gauge recorded a stress of 3MPa during the first fall period. In figure-3.18, Shows graph of the Advance in meters vs Convergence in mm, where X- Distance from goaf edge in meters and Y-Covergence in mm.

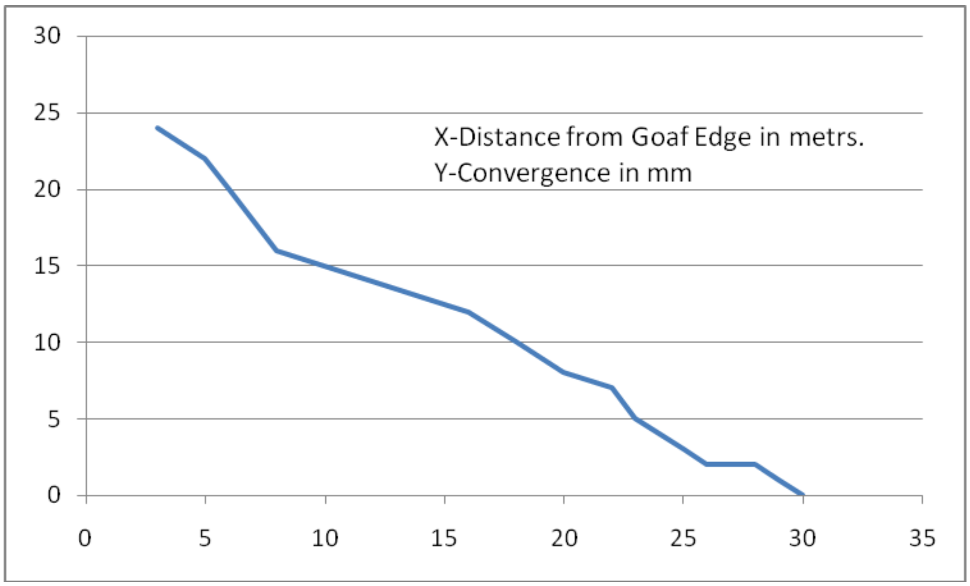


Figure-3.18: Convergence vs. Distance from goaf edge during Depillaring operation

3.10 Summary

It is a difficult to do field instrumentation and monitoring of strata control parameters in and around a productive mine because, generally, such work hampers production interest of the mine. As per observation of roof convergence instrumentation for this study in field monitoring of strata behavior at NCPH Colliery, R-6 Mine is inconclusive. Roof convergence monitoring has important application in roof stability monitoring, particularly during depillaring operations in thick seam working, where roof convergence rates and magnitudes are typically greatest. No geotechnical instrumentation has been used at R-6 Mine before this study.

Monitoring of strata behavior with respect to convergence, we had set remote type of convergence indicators and other instruments such as strain gauge; tall-tale; extensometer; load cell; stress capsules were installed at strategic point in the panel. Empirical relationships to determine the critical and maximum convergence rate for underground roadways, developed from experience in Indian coalfields, suggest a critical and maximum convergence rate of 0.50 mm/day respectively for R-6 mine roadways. A critical convergence rate of 1 mm/day is recommended for the massive sandstone roof at R-6 mine Geotechnical instrumentation is very useful for determining safe convergence rates for junctions and roadways. Further analysis has been done in this respect in forthcoming chapter 4. The following key strata behavior observed during field investigation as per instrumentation plan:

- **Convergence**: Maximum 4mm convergence at the rib pillar and minimum 1mm at 16m away from the point of continuous miner operation for pillar extraction.
- **Strain development/loading on rock bolts**: Strain development was observed at a maximum 3 mm at the pillar side within 6m distance and a minimum 1mm at the pillar side away from 19 m of pillar extraction in continuous miner operation.
- **Displacement of different rock layers**: Displacement of immediate rock layers has found maximum up to 9mm of roof strata at pillar extraction. This is found vary from 3mm to 2.5mm in relation to width of extraction 5.4--6m and 2.4m in working height when extraction was performed along the roof at development panel. After sequencing height and floor during recovery, a slight increase of 3mm in increase of working height up 3 meter and more is noted.
- **Stress developed on pillars/ribs**: When the depth of cover is 75m, stress on the 22m x 22m pillars is noticed up to a maximum of 3Mpa and minimum 1.6 Mpa and varying from 2 to 5 Mpa at the time of stooking or final extraction of pillar. Final pillar extraction reduces the pillar size with the diagonal slicing method inducing stress on the rib to a maximum of 6 Mpa and minimum 2Mpa.

CHAPTER- 4

ANALYSIS OF OBSERVATIONS

ANALYSIS OF OBSERVATIONS

4.0 General

An attempt is made to correlate observed value of convergence with site conditions and characteristics of overlying roof strata at the studied during extraction of coal in a thick seam (6.5 m) by continuous miner with diagonal slicing at NCPH Colliery, R-6 Mine. The design of safe excavation spans promotes safety for underground personnel, mining equipment, services and infrastructure and increases operational efficiency. Excavation spans strongly governs strata behavior, and consequently the design of safe roof spans is also a basis of effective ground support design. Furthermore, the maximum roof span that a coal mine roof can support strongly governs the strata response of the mine roof and the amenability of full-extraction of thick seam by continuous miner with caving mining methods. Sandstone roof in underground coal mining condition is casually known to challenge rock engineers in designing safe excavation spans and predicting roof behavior during depillaring. Specifically, full pillar extraction is difficult to safely implement with sandstone roof conditions due to the large spans required to initiate caving, the unpredictability of caving processes, and the potential for large air blasts in the case of massive roof failures. Qualitative observations and quantitative measurement of strata behaviour was conducted in the S-1 panel during widening of gallery and extraction of pillars.

4.1 Strata behavior

A proper understanding of the influence of strata characteristics on progressive convergence behaviour and support performance is essential for reliable planning, rational support selection and safe operation during extraction of underground coal deposits. During mining of coal seam by continuous miner, excavation is made in the strata, the pre-mining system of forces is disturbed and the energy which is existed in the strata is released and the pressure redistributed itself. The development of pressure on pillars appears to be dependent on three factors.

(a) Depth from the surface.
(b) Area of development.
(c) Ratio of areas of board centers to areas of pillars formed.

Full pressure on pillar due to the weight of overlying strata is experienced much sooner in shallow mines than in deep mines. Strata behavior mainly depends in underground coal extraction by Board and pillar mining method:

- Immediate roof formation
- Method / sequence of winning coal.
- Pillar standing time.
- Extraction height of coal seam.

To study the effect of characteristics of overlying roof rock strata on basis of convergence, the physico-mechanical properties of overlying roof rocks are determined. Observed geo-mechanical properties of roof formations at field investigation site are presented in the previous chapter-3.

Keeping in view of above geo-mining parameter, we had conducted field investigation in NCPH Colliery, R-6 Mine for monitoring of strata behavior with respect of convergences. It was observed that **convergence is a function of time, strata quality, support system, roof span, depth of opening, initial state of stress, time etc.** The roof movement data was analyzed to explain the strata behavior in this field investigation site.

As discussed above, characteristics of overlying roof strata plays an important role on strata behavior and support response during extraction of coal seams. The success of depillaring of coal seams to a large extent depends on this parameter. It indicates the location of roof horizon, from where; the movement of roof strata takes place. Also, the nature and method of mining is interrelated with the movement of strata around the excavation. It is, therefore, quite natural to classify the roof rocks on the basis of their geo-mechanical behavior. Roof rock mass classification is done by different researchers on the basis of different geo-mining parameter. They have developed various empirical models considering different influencing parameters obtained after field and laboratory studies.

Prediction of strata behavior by theoretical analysis becomes unreliable due to the problem of simulation of the real field conditions in mathematical, physical or numerical models. Thus empirical formulation based on the situ measurement of strata behaviors parameter, is an accepted way to estimate the strata behavior. Earlier expressions representing the behavior of opening where usually time-dependence logarithmic or exponential functioned deviated based upon curves fitting to the experimental data obtained from convergence measurement carried out in underground. Convergence of roof to floor, load on supports and bed separation in the immediate roof in the panel S-1 are discussed below.

4.2 Analysis of Convergence

In NCPH Colliery; R-6 mine convergence study has been assess in two segments first during widening of gallery for deployment of continuous miner and secondly study of strata behavior with respect to convergence and as well as support performance.

4.2.1 During Widening of Gallery

The monitoring of the roof deflection has been divided into two parts. The first is the short term dynamic performance during the widening process and the second, the longer term strata behaviour of the 6 m wide roadway with time. With the analysis of all collected data and visual observations of strata after widening of gallery no significant change has measured. The change in shape and increase in displacement values in instruments reading of the gallery occurred in the anticipated sequence in all three deflection profiles. Although not conclusive, this suggests that during this period the 3.0m thick roof beam being monitored remained intact.

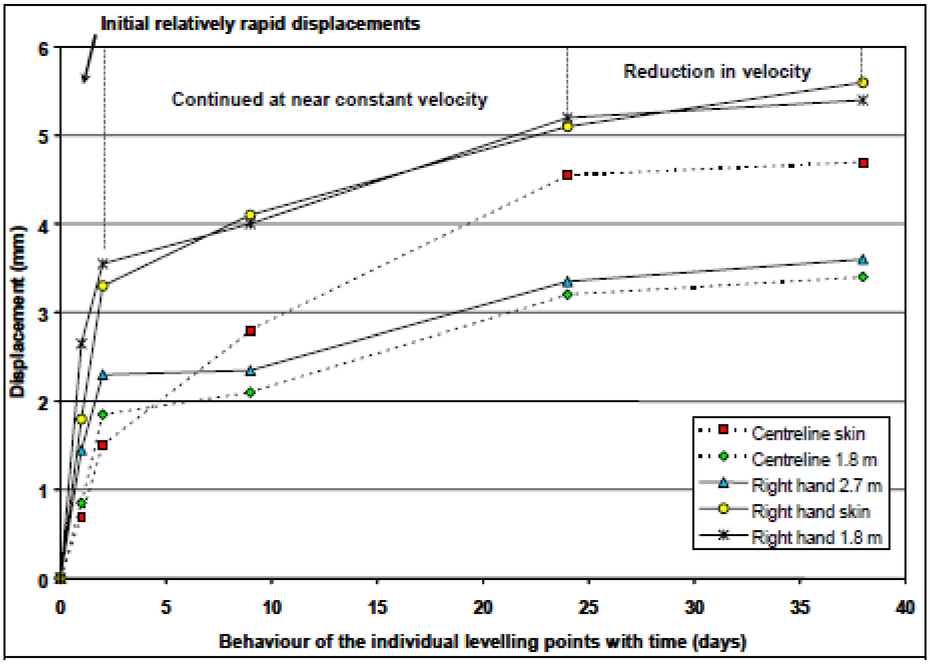


Figure-4.1: Displacement rates as a function of time

Figure 4.1 covers the time period from day one, when the first set of reading were taken in gallery once the width had been opened to 6m, up until the final reading on day 59.The last reading of instruments was recorded to the site on day 72 as the area was about to be finish the widening of gallery in 7 Level east and 6 Level east.

The ultimate aim of this field investigation was to establish the critical roof deformation prior to roof failures during widening of gallery. However, due to competent nature of the roof, it could not be established. Even during the monitoring period no roof falls occurred but small amount of roof spalling notice, where geological disturbance observed visual and maximum 2mm displacements was measured. As an analysis of data it was not possible to try and establish critical roof displacement values for wider gallery (6 mm) width.

4.2.2 During extraction of Pillar

Convergence stations are installed at about 10 m interval along the levels; 1LE, 2 LE, 3 LE, 4 LE, 5 LE, 6 LE and 7 LE in the CM panel S-1. Convergence observations at S9-2LE indicated no perceptible roof movement. About 4 mm cumulative convergence was noticed at this station less convergence may be aborted to the barrier effect.

Convergence was observed at S9-3LE. About 18 mm cumulative convergence was noticed at this station up to the end of 31st Dec'2010. Sounds were observed in 3LE followed by stone fall in pre-shift on 10th August 2011, with convergence of 1 mm only at the station S9 – 3LE.At 2LE, it is observed that station S-7 has the cumulative convergence 22 mm. Maximum rate of convergence of about 2 mm was observed when the station was nearer to the goaf edge i.e., 4 m. Sounds observed and followed by stone fall in pre-shift on 10th Dec'2010. At station S6 in 3LE about 38 mm cumulative convergence was noticed up to the end of 31st Dec'2010. Maximum convergence about 3 mm was observed when the station was nearer to the goaf edge i.e., 3 m. Stone fall has took place on 20th Dec'2010in third shift.

The station C5-4LE was installed on 05th Dec'2010 at a distance of about 16 m from the goaf edge and maintained up to 31st Dec'2010. Maximum of 13 mm cumulative convergence was observed for the last five days as it is very nearer to the goaf edge. Total cumulative convergence at this station is 23 mm (Table-4.1).

Table-4.1: Summary of Convergence Observations

Sl No	Location	Convergence observation				Stone fall details
		Cumulative convergence		Max. convergence change in a day		
		(0.5m)	(2.5m)	(0.5m)	(2.5m)	
1.	7LE	12mm (S 8)	8mm (S 8)	1mm(S 8) on 24-12-2010 GED 3m	1mm(S 8) on 23-12-2010 GED 3 m	Natural fall occurred on 10-12-10
2.	6LE	21mm (S 8)	11 mm (S 8)	3 mm(S 8) on 09-12-10 GED 12 m	2 mm(S 8) on 09-12-10 GED 12 m	Natural fall occurred on 03-08-10, 10-08-11 and on 30-08-11.
3.	5LE	22 mm (S 7)	20 mm (S 7)	2 mm(S 7) on 28-12-10 GED 4 m	2 mm(S 7) on 28-12-10 GED 4 m	Natural fall occurred on 10-12-10.
4.	4LE	35mm (S 6)	28 mm (S 6)	3 mm(S 6) on 30-12-10 GED 3 m	3 mm(S 5) on 30-12-10 GED 3 m	Natural fall occurred on 18-11-10 and on 29-11-10.
5.	3LE	36 mm (S 6)	21 mm (S 6)	3 mm(S 6) on 26-12-10 GED 4 m	3 mm(S 6) on 26-12-10,GED4 m	Natural fall occurred on 01-12-10, 03-12-10 and on 27-12-10.

6.	2LE	23 mm (S 5)	14 mm (S 5)	3 mm(S 5) on 27-12-10 GED 3 m	3 mm(S 5) on 21-12-10 GED 5 m	Natural fall occurred on 01-12-10, 03-12-10and on 27-12-10
7.	1LE	24 mm (S 5)	22mm (S 5)	3 mm(S 5) on 30-12-10 GED 10 m	3 mm(S 5) on 30-12-10 GED 10 m	Natural fall occurred on 12-12-10 and on 30-12-10.
8.	Widening of Gallery (4.5m-6.0m)	6 mm (S 3)	4 mm (S 3)	2 mm(S 3) on 29-08-10	2 mm(S 3) on 29-08-10	No fall

A fall took place after indicating 3 mm convergence on 27th Dec 2010 in first shift. The station S3-2LE was installed on 10th Dec 2010 at a distance of about 22 m from the goaf edge and it is observed that maximum daily convergence recorded when the goaf edge is 8 m from 7th station. Total cumulative convergence at this station is 17 mm. Maximum convergence observed when station nearer to goaf edge.

4.2.3 Bed separation

Bed separation was observed with anchors at a distance of 2.5 m and 0.5 m in the roof in the present panel. Maximum bed separation recorded at 4LE was about 7 mm during 21.12.10 to 20.1.2011 till the station reached goaf edge. In 3LE bed separation of about 10 mm was observed within 0.5 to 2.5 m horizon in the overlying roof near the convergence station S3. 1 mm bed separation was noticed in the horizon of 0.5 to 2.5 m in the overlying strata during 21st to 27th July, 2011. It can be inferred that 3 mm bed separation took place in the immediate roof beds overlying 2.5 m probably at about 7.9 m due to weak clay and carbonaceous shale bands as intercalation between sandstone and coal seam.

4.3 Support behavior

Performance of designed support system mainly depends on strata response and their failure mechanism. Control of failure mechanism of strata is basic principle of support design theories. Characteristic of strata has also play vital role for extraction of coal in a thick seam by continuous miner. In our study analysis shows that following are the main strata characteristic play significant role in support design and response.

- Establishment of overlying strata height with respect to instabilities reached during extraction of coal.
- How these instabilities can be supported and;
- Establishment of strata behavior based on the magnitudes and deformations

Keeping all the above strata characteristic in mind, one of the our objective of study to verify the suitability of existing support system for ensuring safety based on field observations and numerical models. For analyzing the support performance of existing roof bolt in our field site the following two methodologies has adopted:

4.3.1 Visualization of roof bolt

Over the course of the field study (July 2010 to Dec. 2011) very few rock bolts were observed to have failed within roadway excavations. The few observed failures where related to one of the following:

- Rock bolts installed in fault zones (separation of rock around the bolt or failure above the bolt).
- Rock bolts installed in wedges defined by geologic structure (bolt capacity exceeded).
- Rock bolts installed in weak siltstone (separation of rock around the bolt).
- Rock bolts installed in ground subject to weathering (separation of rock around the bolt).

4.3.2 Load on supports

As analysis of observations of load on resin bolts in the CM panel S-1 up to the end of Dec 2011 are presented in Table -4.2. Resin bolts of 21 Tons capacity are set at about 8 to 10 T in majority of the supports in the panel. At station L3-2LE the cumulative load has reached up to 6 T when it was nearly 8 m from the goaf edge, it was installed with a pulling load of 5.35 T at about 19 m from the goaf edge. Maximum daily variation observed was 1 T on 24-04-11 when it was 13 m from the goaf edge. Load cell at station L4-3LE the maximum variation of load was recorded about 6 Tons. At the time, it was nearly 13 m from the goaf edge. Maximum daily variation observed was 2.3 T on 29-12-10 when it was 15 m from the goaf edge. Figure-4.2 shows the graphical representation and explanation of a typical geological profile, support type and roof strata behavior.

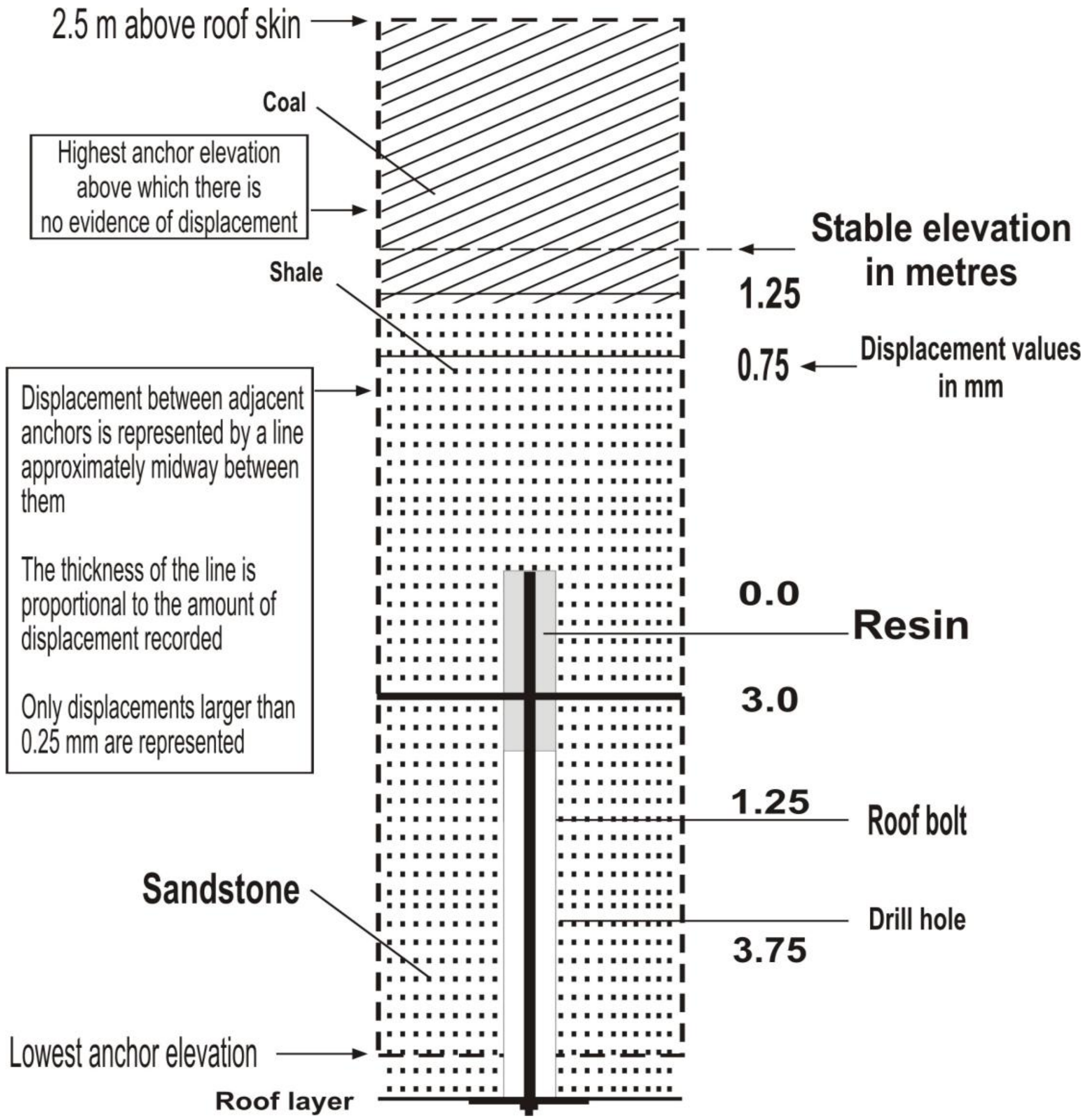


Figure-4.2: Typical geological profile for support type and roof strata behavior

At station L5-2LE about 4.5 Tons variation of load was recorded when it was nearly 3 m from the goaf edge; it was installed with a pulling load of 9.8 T at about 18 m from the goaf edge.

Maximum daily variation observed was 1 Ton 02-12-10 when it was 15 m from the goaf edge. Maximum load observed when station within 10 meters from goaf

Table-4.2: Observation of Load on supports

SL. No	Location	Cum load (Tons)	Max. changed load in a day (Tons)	Stone fall details
1.	7LE	2.957	1.107	Natural fall occurred on 10-12-10.
2.	6LE	2.457	0.870	Natural fall occurred on 03-12-10, 10-12-10and on 30-12-10.
3.	5LE	2.190	0.907	Natural fall occurred on 10-12-10.
4.	4LE	0.300	0.127	Natural fall occurred on 18-11-10and on 29-11-10.
5	3LE	4.584	1.000	Natural fall occurred on 01-12-10, 03-12-10and on 27-12-10.
6	2LE	3.947	0.783	Natural fall occurred on 01-04-11, 03-12-10and on 27-12-10.
7	1LE	4.120	1.237	Natural fall occurred on 12-12-10 and on 30-12-10.
8	Widening of Gallery (4.5m-6.0m)	1.057	1.033	No fall

Maximum rate of convergence reached up to 4 mm per day at S4-4LE. However, it remained within 1.5 to 2 mm/day for the other stations located within 40 m from the diagonal line of extraction. The maximum load on support remained within 12 Tons in most of the cases within 10 m from the goaf. Irregular trend of variation of the load on support may be partially attributed to the disturbance of the supports occasionally by the movement of strata. In general, continuously increasing trend with a maximum rate of 0.5 to 2 Tons/day was observed. The supports were loaded up to 35% of their capacity without any abnormal behaviour of the supports till the end of Dec 2011.

4.3.3 Roof Bolt Pull Tests

In order to ensure that the **Roof Bolt** anchorage capacity matched the bolt strength, **Roof Bolt** pull tests were carried out throughout the mine. Random pull tests were carried out on production bolts as well as on bolts specifically installed for the purpose of pull tests. Several mechanical anchor and Split Set bolts were tested in addition to the rebar bolts. A total of 54 pull tests were carried out during operation of continuous miner in S-1 panel, NCPH Colliery, R-6 mine, Seam No.3. Details of the test are presented in Appendix 3. Several standards are in common use to evaluate if a rock bolt pull test is acceptable or not; these standards are as follows:

- Acceptable if less than 3.18mm of anchor displacement occurs at a 7.25 tonne load (Mark et al, 2000b).
- Acceptable if the bond stress at failure exceeds 5 MPa Failure is considered to have occurred when the slope of the bond stress versus anchor displacement curve drops below 0.75 MPa per mm (British Coal, 1992).
- Acceptable if the bolt can be loaded to the yield strength without sustaining unrecoverable deformation (Cullen, 1989).

Random pull tests were carried out on 60 standard production bolts. To minimise the need to replace tested bolts most tests were carried out to a maximum load of 13 tonne, which was the yield load of the rebar. The results from these tests are summarised below:

• The average failure load was 18 tonne (excluding failures at the nut).

• The minimum failure load was 8.5 tonne (failure at the rock/resin interface).

• The average load at which nut failure occurred was 15 tonne.

• The minimum load at which a nut failure occurred was 9.8 tonne.

In 20% of the random pull tests, failure occurred by the nut stripping, which clearly indicated a problem with the bolt manufacture. This problem was brought to the attention of the bolt supplier and the problem was rectified by changing the way that the threads were formed.

The tests were carried out in general conformance with the ISRM standards. The tests monitored the deformation as well as the load applied to the bolt. Deformation due to stretching of the rebar was backed out of the calculations using the elastic modulus of the steel. Beyond the rebar yield load (13 tonne) this calculation could no longer be performed, as stretching became non elastic. As the primary interest of these tests was to establish a bond capacity, it was desirable to have the bond fail below the steel yield load. To accomplish this, the resin encapsulation length was limited to between 0.22 and 0.35m; these types of tests are often referred to as "short encapsulation" pull tests.

The first short encapsulation tests were carried out to evaluate the anchor capacity in competent siltstone. The length of resin encapsulation was 0.305m. In all but one test the yield point of the steel was reached (13 tonne). This indicated that 0.305m of resin encapsulation provided sufficient anchorage to reach the roof bolt yield load. One test failed at a 3 tonne load due to improper resin mixing.

Pull tests were carried out at the following intervals: 1 day, 32 day, and 65 day. Results for the tests conducted at 65 days are presented graphically in Figures 5.1 and 5.2. The results indicated that there was no time dependent loss of anchor capacity in weak siltstone that was dry. The results also indicated that the average resin anchor capacity is 24 tonne/m resins. The optimum encapsulation length, which is the minimum resin length required to reach the ultimate strength of the rebar, is therefore 0.877 m. The pull test acceptance criteria discussed previously were met for most of these tests.

Pull tests of production bolts installed in sandstone that was wet were also carried out. Yield failure, related to resin-rock anchor slippage, began to occur at approximately 8 tonne. The average failure load was only 15.9 tonne. These tests did not meet any of the pull test acceptance criteria. These results indicated that rock bolts installed in weak siltstone might not be acceptable if moisture is present. The following recommendations were made to deal with this problem:

- Install longer bolts that anchor in competent rock beyond the weak siltstone.
- Design the support based on a peak load of approximately 19 tonne, which would require that bolt spacing to be reduced.

Despite its shallow depth the NCPH Colliery, R-6 Mine was relatively dry. Within mine roadways water was only a problems in the vicinity of major faults.

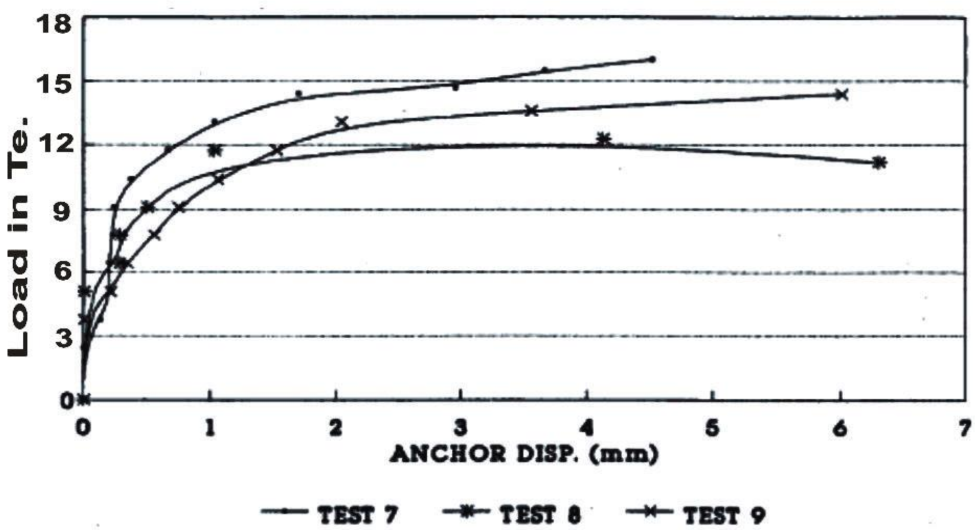


Figure-4.3: Load versus Displacement. Short Encapsulation Pull Test Results for Bolts Installed in sandstone after 65 Days

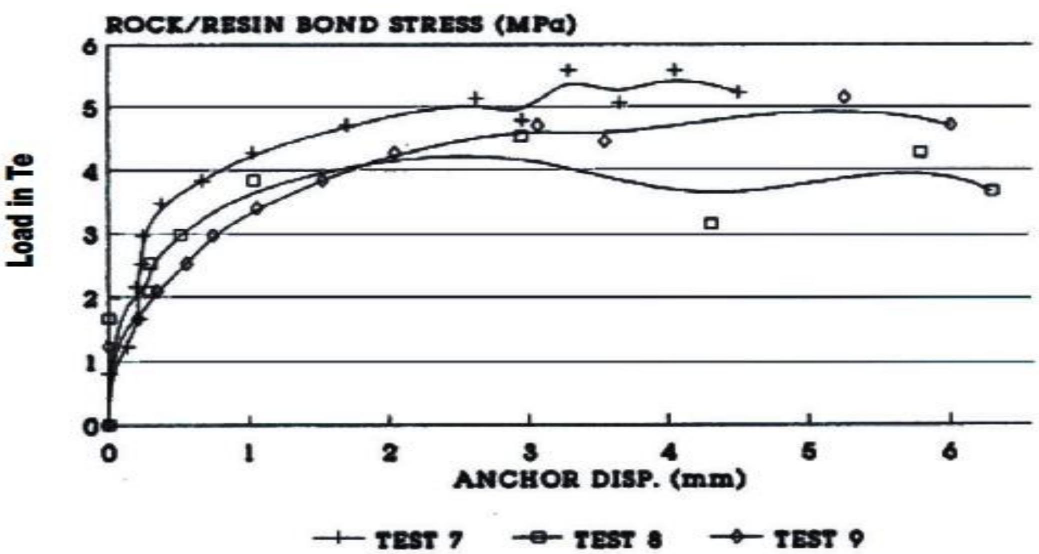


Figure-4.4: Rock/Resin Bond Stress versus Displacement. Short Encapsulation Pull Test Results for Bolts Installed in sandstone 65 Days

Bolt tension measurements were carried out at the same time as many of the pull tests. Very little tension bleeds off was found to occur with properly installed bolts in competent siltstone. Significant tension bleed off (up to 50%) occurred on bolts installed in sandstone.

It was concluded that bolt tension could be used to assess the quality of the rock bolt installations. Where tension bleed off greater than 20% occurred, the bolt may be improperly installed or the anchor at the end of the hole, or at the collar, may be yielding. An anchor problem would be suspected if tension bleeds off continued to occur after the bolt was retorqued.

A total of five Split Set Bolts and three mechanical anchor bolts were tested. All tests were carried out in competent siltstone. From these tests it was determined that the average yield load was 2.9 tonnes per metre length for the Split Set bolts, the average anchor failure load for mechanical bolts was 8.3 tonnes. The results of the tests are summarised in Appendix 3.

Compared to the point anchor resin bolt the capacity of the Split Set bolts and mechanical anchor bolts is low. The use of Split Set or mechanical anchor bolts was not recommended where the roof rock consisted of sandstone.

Drill Hole Size: The size of the drill hole, relative to the size of the bolt, is a critical factor in the quality of a resin anchor rock bolt installation. If the annulus between the drill hole and bolt is too large the resin may not be mixed properly and may be "spun away" from the anchor zone. The standard drill bit used at the NCPH Colliery, R-6 Mine was a 25 mm cross bit. Borehole callipers were used to determine the finished dimension of the drill hole. Measurements were taken in different rock types throughout the mine. The average drill hole diameters 26.2 mm for the sandstone rock types.

During observation of study the primary cause of the decrease in anchor capacity with increasing hole size is improper mixing of the resin and/or resin being spun away from the anchor point. These problems increase as the size difference between the bolt and the drill hole increases. "Glove fingering" of the resin occurs when the resin cartridge is punctured but not properly shredded and mixed. This phenomenon usually occurs where the size of hole is more than 10mm larger than the size of bolt being installed, or when the rebar is pushed either too quickly or too slowly through the resin (Ulrich et al 1991).

Rock Bolt Load: Total ten numbers of load cells was placed on a rock bolt to study the load history. The bolt was installed with the load cell approximately 40m back from an advancing gob. The load cell was left in place until just before it passed into the gob, at which time it was recovered. The load history is shown in Figure-4.5. As can be seen, the load on the rock bolt did not change from the time of installation to the time of removal (just prior to passing into the gob). Since there was no increase in load on the bolt, it can be concluded that there was no deformation in the roof rock even as the gob approached. This also infers that there was no change in the induced stresses in the immediate roof.

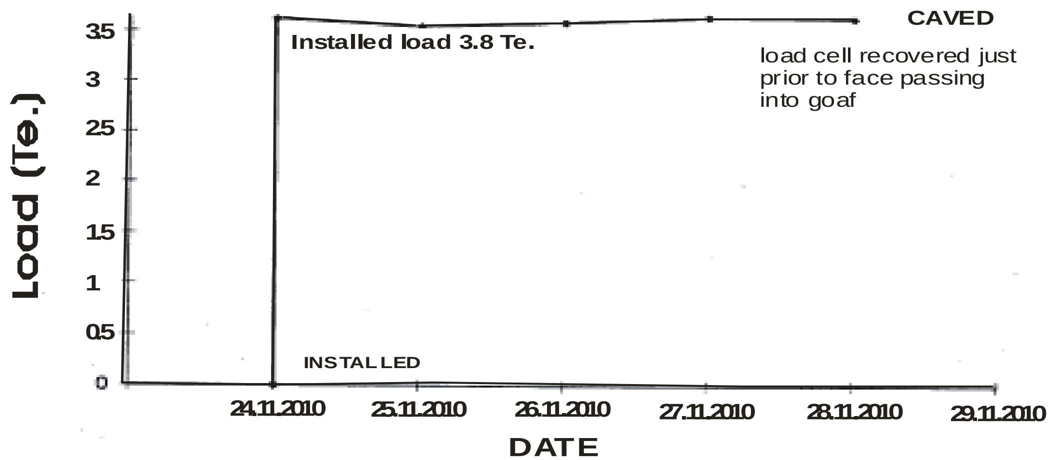


Figure-4.5: Load History on Rock Bolt Installed in depillaring Area

Full Column Resin Rock Bolts: Full column resin bolts have several advantages over point anchor bolts:

- Filling the hole with resin reduces moisture ingress and deterioration of the rock.
- The resin provides anchorage along the full length of the bolt including the collar rock, this helps prevent separating around the collar.
- If the rock around the collar deteriorates a full column rock bolt remains effective. With point anchor bolts once the rock around the collar is lost the bolt is essentially ineffective.
- Full column resin increases the bolt stiffness and improves shear resistance.

At the NCPH Colliery, R-6 Mine, deterioration of rock around the collar of the bolts was found to be a significant problem in the weak siltstone. Installation trials of full column resin bolts were carried out. During the trials, 30% of the bolt installations failed due to bending of the rebar as it was being inserted into the hole.

It is common industry practice to install 1.8m long, 20mm diameter bolts into 27mm diameter holes, with full resin encapsulation shows in figure-4.4. Resin manufacturers and rock bolt manufacturers were brought in to examine the installation problem at the NCPH Colliery, R-6 Mine. Practically observed that the Roof Bolters were not capable of drilling a straight enough hole or pushing the rebar through the resin in a straight enough line to assure trouble free full column resin installation.

4.4 Rock Bolt Pull Tests for Efficient Length

Rock bolt pull tests conducted at the NCPH Colliery, R-6 Mine intended to evaluate alternative, less intensive and costly ground support options amenable to the good roof conditions in NCPH Colliery, R-6 Mine. His recommendation for primary ground support were as follows:

Bolt spacing: 1.5 m x 1.5 m

Bolt length: 1.5 m minimum

Bolt type: point anchor resin or mechanical (Quick Setting cement Capsule)

Bolt capacity; 9 tonnes

As per our field investigation, "The use of the mechanical bolt option should be successful within the sandstone support of the NCPH Colliery, R-6 Mine for a test production panel. The specifications must be reviewed and assessed in terms of bolt strengths/plates/shells/lengths with respect to patterns similar to other panel for S-1 panel in this mine. Quality control of the bolts must be assessed and tested on an ongoing basis. They are not to be used in structurally controlled/faulted ground." Before and during this work, 1.8-meter long point anchor rebar were installed on 1.2-meter centers, strongly believed to be excessive during the time. In NCPH Colliery, R-6 Mine performed 3 standard pull tests on 1.8-meter long, Grade - Fe 415 IS: 1786, point-anchor resin rebar and 4 x 1.8-meter long, C1060 Grade #5 mechanical bolts. A total of 29 pull tests were conducted on forged-head bolts of bolt lengths 1.2, 1.5 and 1.8 meters. About 1 meter (66% of bolt length) of resin anchorage was achieved for all tests. A loading rate of about 4.5 tonnes/min was maintained during all tests. A summary of the pull test results for full-column resin forged-head bolts of 1.2-meter length is shown below in Figure-4.6: No bolts pulled out or failed for all ten tests. The yield load ranged between 12 and 14.5 tonnes at 7 mm displacement. The maximum displacement of 32 mm was achieved at a load of 21 tonnes.

A summary of the pull test results for full-column resin forged-head bolts of 1.5-meter length is shown below in Figure-4.7. No bolts pulled out or failed during testing. The yield load ranged between 12 and 16 tonnes at 8 to12 mm displacement. The maximum displacement of 27 mm was achieved at a load of 20 tonnes. A summary of the pull test results for full-column resin forged-head bolts of 1.8-meter length is shown below in Figure 4.21. No bolts pulled out or failed for all tests. The yield load ranged between 14 and 18 tonnes at 5 mm displacement. The maximum displacement of 25 mm was achieved at a load of 19 tonnes.

The short-encapsulation pull test achieved a yield load of 18.5 tonnes at 65 mm displacement. The anchorage length was 26mm corresponding to a minimum anchorage factor of 50 tonnes/m in massive sandstone. Excellent anchorage capacity is indicated for forged head bolts in the immediate roof.

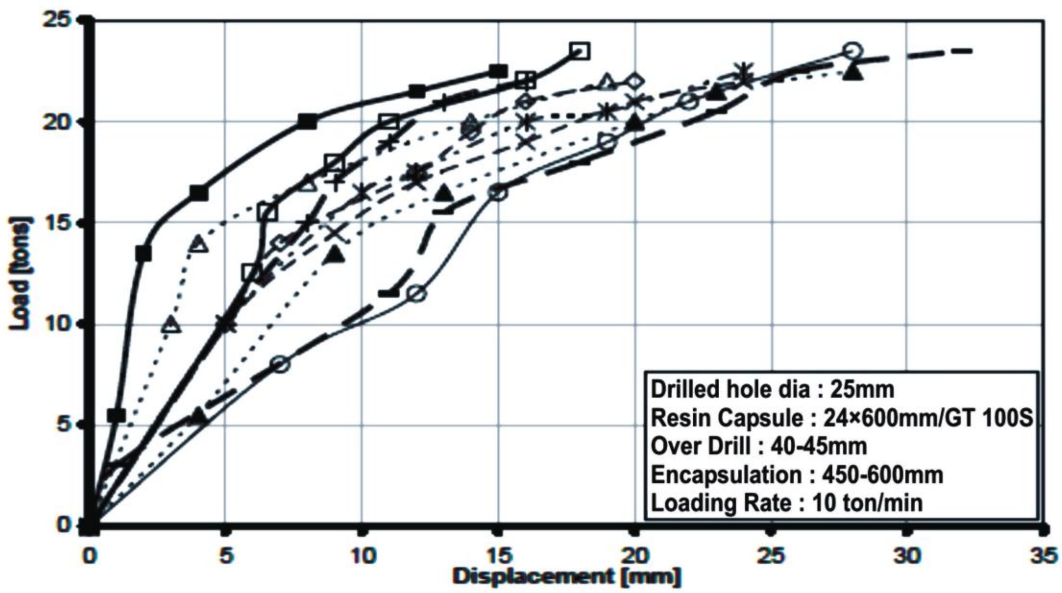


Figure-4.6: 1.2-meter Long Full-column Forged-head Rock Bolt Pull Test Results

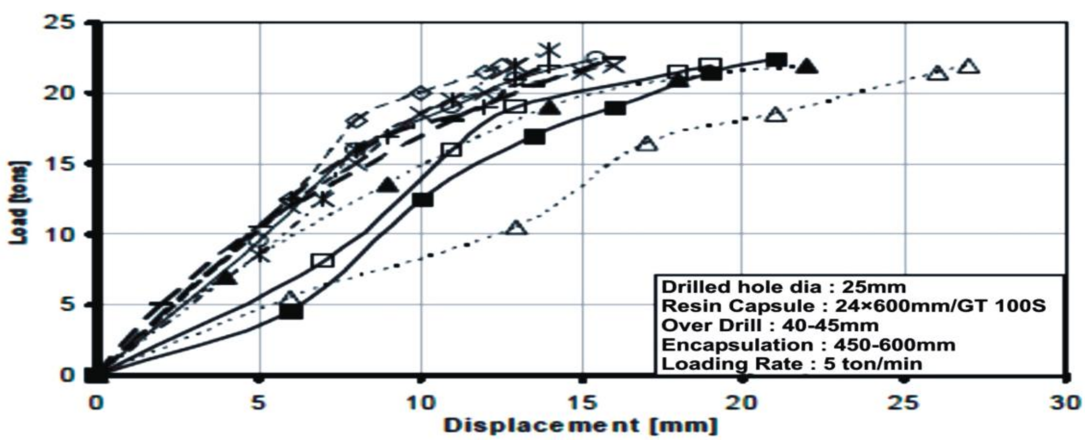


Figure-4.7: 1.5-meter Long Full-column Forged-head Rock Bolt Pull Test Results

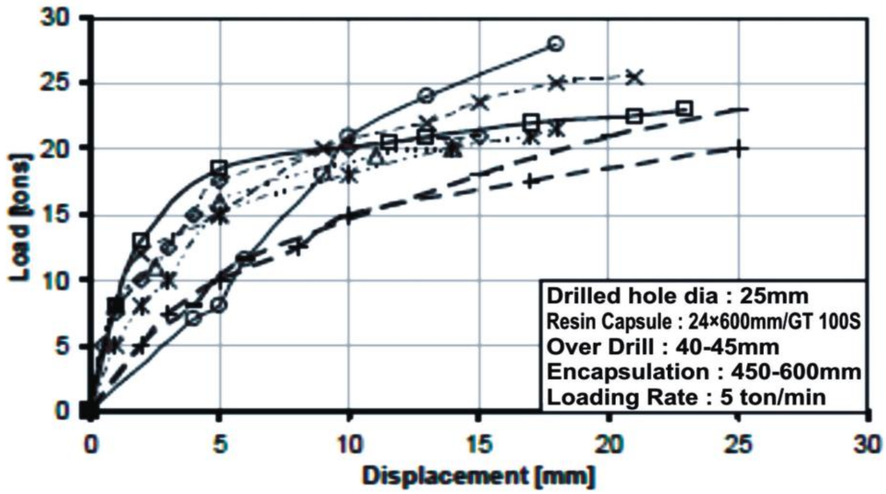


Figure-4.8: 1.8-meter Long Full-column Forged-head Rock Bolt Pull Test Results

Four pull tests (No.9-12) were conducted on tensioned rebar of 1.8-meter length. Resin anchorage was set at 0.38 meters for all tests. A summary of the SEPT results for tension rebar bolts of 1.8-meter length is shown below in Figure 4.9. The yield load varied from 13 to 18 tonnes at 5, 10 and 15-mm displacement. Maximum displacement was 44 mm at about 18 tonnes. This part of our study main intended to determine the safest, productive and cost effective ground support strategy by examining the support density, rock bolt length, and optionally, type to determine the safest, most productive and most cost-effective ground support strategy, which was done by examining the support density, roof bolt length and, optionally, the type of permitted ground support for the NCPH mine of South Eastern Coalfields Limited (S.E.C.L.). This can be also useful in other similar geo-mining conditions in different coal fields in India for mass exploitation of underground coal deposits. Currently, the ground support system for 6m roadways consists of 19 to 22 mm diameter, 1.8m-length, point-anchored resin rebar on 1.2 m centers with Bureau of Indian Standards(BIS) norms grade-bearing plates with no screen.

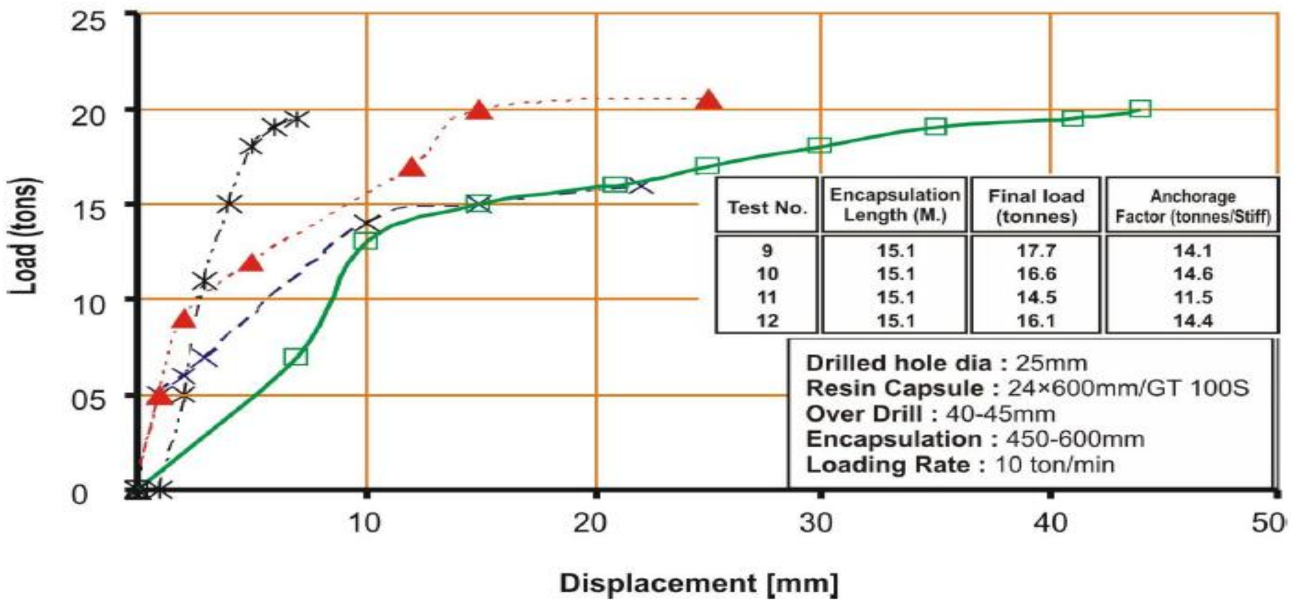


Figure-4.9: Long Tensioned 1.8-meter Roof Bolt SEPT Results.

4.4.1 Comparison of the Existing Support Design Methods to Experience at the NCPH Colliery; R-6 Mine

After the determination of rock mass support design methods in existing support system of bolting and this was used to determine their commended support for a 6m wide excavation in sandstone at a depth of 100m. The rock properties used in the design were those for competent sandstone as found at the NCPH Colliery, R-6 Mine. The results are tabulated in Table-4.3.

Table-4.3: Support Design for 6m Wide Excavation in Competent sandstone

DESIGN METHOD	BOLT TYPE	BOLT LENGTH (m)	BOLT SPACING (m)	BOLT CAPACITY (tonne)	OTHER SUPPORT COMMENTS
CMRI-Geo-mechanical classification	Roof bolt with quick setting cement capsules	1.5	1.2	6	TMT ribbed bolt of 22 mm dia with fos = 1.1
Support used at the	resin - point anchor	1.8	1.5	12	TMT ribbed bolt of 22 mm dia with

NCPH Colliery, R-6 Mine					fos = 1.3
Numerical Analysis by FLAC -3D	Roof bolt with resin capsule	1.8	1.5	12	TMT ribbed bolt of 22 mm dia with fos = 1.2

Almost all the above consider support design methods overestimated the support levels that were found to work successfully at depths between 86m to 100m at the NCPH Colliery, R-6 Mine because empirical assessment based on drilling and blasting whereas continuous miner technology has efficient for strata consistency. In terms of providing a safe excavation it appears that any of the methods would be acceptable.

However, using a method that overestimates ground support will unnecessarily increase the production costs. The numerical assessment method was the only method that recommended ground support levels that were consistent with the levels used at the NCPH Colliery, R-6 Mine.

The main shortcomings of most of the existing rock mass support design tools are that they do not consider the stress regime and/or they were not actualise for use in technology based coal mines. For example:

• CMRI-Geo-mechanical classification method was designed exclusively for coal mines in drilling and blasting operation; however, the data set used to develop the method was obtained primarily from mines operating at depths greater than 100m, and no account is taken of mining technology utilisation.

• The Numerical Analysis methods were developed for use in Mining & tunneling applications based on rock mass characterization.

4.5 Summary

Excavation stability at shallow depth during depillaring in thick seam coal mines should primarily consider geologic structure and the rock mass. Support design should be based on both an empirical rock mass method as well as an analytical geologic structure method. This study has shown that most of the rock mass support design methods unmatched the support requirements at technology utilization and geo-mining condition; the one exception is the numerical analysis method. The FLAC computer program was found to be suitable for determination of support for geologic structure.

Where the depth of cover is less than 100m the induced stresses are expected to be low, and stress induced failures are not expected to be a significant problem. However, it must be recognised that the affects of stress are proportional to the strength of the rock mass; the potential for stress related instability is greater in weaker rock/ weakened by weathering processes.

As the depth of cover decreases the stresses over the top of an excavation will decrease and may become tensile. This situation favours structurally controlled gravitational type failures such as bed separation, beam bending, sagging, and block type failures. Gravity induced failures are expected to be the most prevalent type of failure at shallow depth coal mines where well defined geologic structure is present. The FLAC computer program developed by Itasca Consulting Group, USA was determined to be an excellent tool for visualization of wedges and determination of support requirements to secure the wedges.

This study has shown that most of the existing rock mass support designed methods verified with support requirements at present geo-mining conditions and extraction of coal in a thick seam (6.5 m) by continuous miner with diagonal slicing. There are three reasons for this:

- The methods do not generally consider depth of cover or suitability technology.
- Most of the methods were developed for uses other than continuous miner technology at coal mines.
- Most of the methods were developed using data from excavations greater than 100m.

Combination of empirical and numerical assessment was found to match the support levels in use at the NCPH Colliery; R-6 Mine where the continuous miner is working at depths between 86m to 100m. After analysis of rock mass support design method that was verified the suitability of existing support system for ensuring safety based on field observations for use in R-6 Mine that also considers the thick seam strata behavior during depillaring. Support design based on most of the other rock mass methods evaluated should provide a safe excavation but not a cost effective excavation.

Rock bolts consisting of point anchor rebar were found to be a suitable means of strata control in competent sandstone. In sandstone that is dry the average resin bond capacity was determined to be 24 tonne/m. If 22mm diameter grade 60 rebar is used the optimum resin encapsulation length is 0.88m; this is the resin length required to hold the ultimate strength of the rebar (21 tonne).

In weak a stratum that is wet the resin bond capacity was found to be less than 12 tonne/m. Yield at the resin-rock interface began at loads as little as 4 tonne. The ability of resin anchored rebar to provide adequate anchorage in weak wet siltstone is questionable. Where weak strata is present the bolt length should be increased to anchor into competent rock; if this is not possible and there is a potential for the rock to become wet the support design should utilise a design load of 4 tonnes per bolt. Consideration must also be given to the problem of separation and weathering type failures that commonly occur in weaker rocks.

This study has shown that rock bolt tension can be used for quality assurance testing at the NCPH Colliery, R-6 Mine. Where tension bleed off exceeds 20% either there is anchor slippage or the bolt was not installed properly. Bolt tension and load were found to remain essentially unchanged throughout the life of stable excavations. This finding is consistent with the results of the convergence study reported in this study.

The key analysis drawn from this field investigations include:

1. Roof bolt corrosion has been limited to thin surface rust films on the installed plates and exposed rebar. Case history reflection and empirical estimate references for roof bolt longevity strongly suggest that corrosion will not measurably affect the rock bolt strength until 10 years post-installation.
2. The elastic beam theory of rock bolt support function is the best general model for the NCPH mine of S.E.C.L. for sandstone roof. Where vertical discontinuities extend through the bolted roof thickness, beam theory may be more appropriate.
3. Wedge support analysis, facilitated with modeling software, has excellent application at the NCPH mine of S.E.C.L. for assessing the capacity of the primary roof support to stabilize roof wedges.
4. Pyramid idealizations of intersection wedge geometry are most appropriate for massive sandstone roof.
5. A critical wedge was determined to be defined by joints angled at 30 degrees above the roof line and a slab of rock formed with a joint angled at 18 degrees based on sandstone roof supported with 1.8m point-anchored rock bolts on 1.5m centers. The corresponding critical angles for wedges and slabs supported with 1.8m point-anchored rock bolts on 1.5m centers are 27 to 30 degrees, respectively. Wedges and slabs formed by joints angled more steeply will require supplemental support assessment.

6. Empirical estimates of support requirement for the NCPH mine roof suggest a conservative range of rock bolt lengths and pattern support spacing, and a greater support load density and anchorage depth that have successfully been provided by the current installed ground support. Methods based on beam theory require excessive bolt lengths of 2.4m for 1.5m center-to-center bolt spacing.
7. The analytical support predictions prescribe 1.8m for split and 2.4 for original gallery long roof bolts, and point-anchored rebar on 1.5m center-to-center spacing a design well suited to the roof conditions in NCPH mine of S.E.C.L.
8. Our field observations recommend 1.8m-long roof bolts on 0.75m and 1.5m center-to-center support spacing best suited for NCPH mine of S.E.C.L.
9. Resin roof bolts are finding increasing applications in underground coal mines, particularly un-tensioned, forged headed rebar, with lengths of 1.8m being the most accepted.
10. The 1.8m-long, 22mm-diameter grade BSI mechanical roof bolts on 1.5m centers are recommended as primary roof support based on pull tests and the resulting bolt minimum bolt capacity of 20 tonnes at 50mm displacement. A plan of quality control of strata management with bolt testing is also established for monitoring within the first three months of usage at a minimum rate of 10% of installation or 50 bolts per month, whichever is higher.
11. Reviews of empirical and analytical roof support design methods recommend either 19-tonne capacity, 1.8m minimum length point- anchored resin rebar, or mechanical roof bolts on 1.5-meter centers to serve as roof supports for the NCPH mine.
12. It is our knowledge that it is a good match between the sandstone roof and mechanical rock bolts based on underground observations; however, the authors/researcher have cautioned that quality assessment and control must be part of the application, and that they would not be appropriate in structurally controlled/faulted ground.
13. Forged head bolts performed very well in underground pull tests. The average yield load and corresponding displacement for rebar lengths of 1.2, 1.5 and 1.8m are shown in Table-4.4. The bond strength from a single SEPT (Standard Encapsulation Pull Test) on a 1.8m length was observed at 50 tonnes/meter. The average bond strength of SEPTs on a 1.8m torque tension rebar was 45tonnes/meter at an average yield load of 15.5tonnes.

Table-4.4: Summary of Pull Test Results – Forged Head Bolts

Bolt's Length (m)	Avg. Yield Load	Avg. Displacement
1.2	13	7
1.5	14	10

Based on the investigations itemized above, advices for ground support at the NCPH mine are summarized in Table-4.5.

Table-4.5: Strata Support Design – NCPH Colliery, R-6 mine of S.E.C.L.

TYPE A	
Roof bolt material	Tor-steel (IS: 1570/IS: 226-1975)
Yield load/Capacity (tonnes)	19
Bolt type	Fully grouted forged-head resin rebar
Resin capsule	24 x600mm /GT 100S resin or equivalent
Length (m)	1.5
Bolts per row	4.00
Rib-to-rib-side bolt spacing (m)	0.75
Bolt-to-bolt spacing (m)	1.50
Row spacing	1.50
Plate	MS- IS: 226-1975 SZ 150mmTH:6mm
Screen	Not required
TYPE B	
Roof bolt material	Tor-steel (IS: 1786-1985/IS: 226-1975)
Yield load/Capacity (tonnes)	24
Bolt type	Mechanical roof bolt w/ hardened steel dome washer plate
Resin capsule	24 x 600mm /GT 50S resin or equivalent
Length (m)	1.8
Bolts per row	4.00

Rib-to-rib-side bolt spacing (m)	0.75
Bolt-to-bolt spacing (m)	1.50
Row spacing	1.50
Plate	MS- IS: 226-1975 SZ 150mmTH:8mm
Screen	Not required
Table-4.5(a):Supplemental support: Faulted/Geological Structure Controlled Ground	
Passive support	Cable bolt or rope stitching for deeper suspension
No.9 Gauge wire mesh	Roof sagginess control
Posts	Roof span reduction/support

The Type B support system recommendation is based on the presented results of other researchers and requires updating to before implementation in the NCPH mine of S.E.C.L or any other mine of the same geo-mining conditions. The material properties of currently available mechanical bolts, including the anchorage capacities of the capsule, should be tested thoroughly before the Type B standard is adopted into the NCPH mine's primary ground support plan. The Roof bolt testing procedure has implemented, as recommended by DGMS Circular, Tech Sapicom 3 of 1993 and 6 of 1996 should be followed until a site-specific procedure is developed at the NCPH mine for mechanical roof bolts. Supplementary supports for face spalling control, passive support roof stitching with wooden lagging and posts for rib and roof support should be reviewed for inclusion on a case-by-case basis or according to approved design rules. Strata support design at the NCPH mine of S.E.C.L for mass exploitation of underground coal deposits are best suited applicable in continuous miner operation panels as outlined in Table-4.5.

CHAPTER-5

RESULTS AND DISCUSSIONS

RESULTS AND DISCUSSIONS

5.0 General

The field experimental studies furnished qualitative and quantitative information about the strata behavior with respect to convergence and verify the suitability of existing support system for ensuring safety during extraction of coal in a thick seam (6.5 m) by continuous miner with diagonal slicing. The numerical model studies supplemented the data on field investigation for ensuring suitability of roof bolt support system at present geo-mining conditions of NCPH Colliery R-6 Mine. The mechanism of the tensioned bolts is that the tensioned bolts support the bolted strata by reinforcing the first 0.60-1 m of the immediate weak roof and provide a high resistance to subsequent roof displacement through the pre-tension. Due to simplicity and convenience of the fracture mechanics approach, its results having inherent assumption of zero crack thickness, can be validated for different thickness of seams. For suitable modification of existing support system added through verification of numerical model studies. Results of different studies and comparison of field observation with numerical and analytical studies on support system are discussed below.

5.1 Strata Behavior

Maximum stress over pillar / stook / rib and convergence of opening around depillaring working was found at the time of first major falls during extraction of 2-3 rows of pillars in the entire investigation panel S-1. Extraction at the time of roof falls.

The trend of convergence combining all the data showed no clear trend for warning of roof falls on the basis of cumulative or rate of convergence. However, it can be said in general terms that cumulative convergence exceeding 18 mm would have possibility of 0.8, while the rate of convergence exceeding 4mm/day can indicate 60% of the roof falls for the instruments located within 20 m from the line of extraction.

5.1.1. Convergence

The results of the convergence in S-1 experimental panel showed similar trend of decreasing convergence value / amount with increased distance from line of extraction towards the solid pillars, which is expected. Similar trend was reported by many investigators with a maximum convergence in the range 5-25 mm near goaf edges during depillaring in seam thickness of 7.5 m (CMRI, 1987). It should be noted that the convergence in these cases was measured upto extraction line only, without any attempts for monitoring the convergence in the goaf. Salient findings of the convergence during depillaring include Maximum of 2-4 mm convergence upto extraction of a row of pillars. This can be attributed to the presence of less roof span in the goaf and ribs without any spalling. These results are comparatively less than the observations in 10-12 m high workings (NIRM, 1997, 1999), most probably due to roof bolting of the roof in the existing systems.

Critical Convergence Rate: Critical roof deformation levels and responses need to be identified based on the rock mechanical conditions and deformation characteristics at the mine site. Measured deformations will be a contribution of deflection of the roof skin and internal deformation within the roof rock mass. Roof deflections of roadways can be predicted by conventional gravity loaded beam theory as the length of roadways exceeds twice their width. The maximum deflection formula is shown below:

$$\eta = \frac{\rho g L^4}{32 E t^2} \qquad \text{……………………} \ (5.1)$$

Where: η =Maximum deflection (m)

ρ = density (kg/m^3)

g = gravity (m/sec^2)

E = Modulus of Elasticity (N/m^2),

t = thickness of layer (m)

L = span width (m)

The amount of roof sag is proportional to the fourth power of road width. This means that small increases in roadway width translate into significantly larger maximum roof deflections. The effect is more pronounced for intersections where deflections are theoretically about 4 times as large. Intersection span increases due to rounding of pillar corners that are rounded for machine travel (turn-outs) or rib spalling require constant vigilance. In 2009, there were 1,985 non-injury reportable roof falls. More than 71% of these occurred in junctions, despite the fact that intersections probably account for less than 25% of all development underground (Mark et al.

2001). Equation (5.1) predicts maximum deflections of 1.8 to 5.1 mm for 6-meter width roadways and 7.0 to 19.5 mm for intersections in the R-6 mine.

Ghosh and Ghose (1992) attempted to establish a relationship relation between the maximum ground movement in a roadway with the rock mass rating, the roadway width and the rock dry density and presented the following relation based on case histories from eight different coal mines in India:

$$C_m = 40B^{0.5}\gamma^{0.3}(1 - R/100)^3 \quad \ldots\ldots\ldots\ldots\ldots\ldots (5.2)$$

Where: C_m = maximum ground movement (mm)

B = roadway width (m)

γ = rock dry density, (kg/m^3)

Equation (5.2) predicts a maximum ground movement of 3.5 mm for R-6 mine sandstone roof in 6 m wide roadways. The rate of movement in the immediate roof has been shown to be a reliable measure of roof instability. Maleki (1988) proposes the rate of movement is favorable to other metrics of instability because a) the rate does not depend on the entire history of roof movement and b) it indicates a change in the stability of the whole mining system. Van Der Merwe (1998) reported on three common displacement-time behaviors for mine roofs as illustrated in Figure-5.1. In this figure, curve (a) represents stable roof requiring monitoring at long intervals, (b) acceleration, typical of imminent failure, and (c) steady deformation, where failure occurs when the maximum magnitude of displacement is reached.

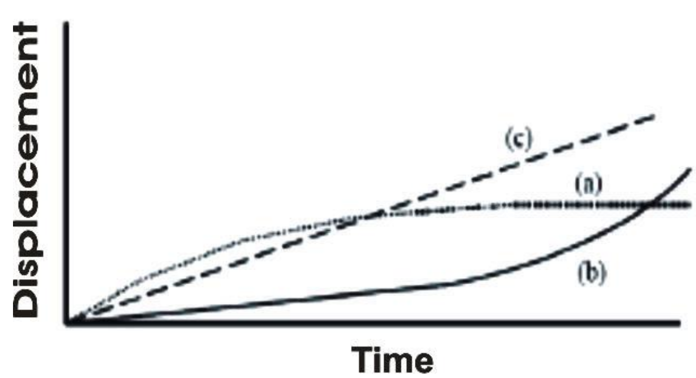


Figure- 5.1: Displacement vs. Time Behavior of Roofs (After Van Der Merwe, J.N., 1998).

Ghosh and Ghose (1995) instrumented the intersections of several underground coal mines in India to develop a relation between the critical convergence roadway width and the rock dry density. Equations-(5.3) and (5.4) show the empirical formulas they developed for critical convergence and maximum convergence velocity respectively:

$$V_R = 2.25B\left(\frac{\Gamma}{1000}\right)^{0.66}\left(\frac{100-R}{100}\right)^{6} \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ (5.3)$$

Where: V_R = critical velocity (mm/d)

B = roadway width (m)

R=Rock Mass rating

Γ = rock dry density, (kg/m3)

$$V_{R\max} = 3.3B^{0.55}\left(\frac{\Gamma}{1000}\right)^{0.36}\left(\frac{100-R}{100}\right)^{3.3} \quad \ldots\ldots\ldots\ldots..\ (5.4)$$

Equations-(5.3) and (5.4) are based on roadways of width 3.0 to 4.8 meters and rock mass ratings between 19 and 50%. These formulas predict a critical convergence and maximum convergence velocity of 0.018 mm/day and 0.23 mm/day respectively for the R-6 mine roadways. Pakalnis (2009) assumes a critical convergence rate of 1mm/day for consulting work in hard rock mines. Cullen (2002) found a critical convergence velocity of 0.03 mm/min (432 mm/day) for the 4LE/14DN mine based on instrumented studies. These results indicate the critical convergence velocity for a massive sandstone roof is more than two orders of magnitude less than for a weaker formation of shale roof.

Convergence measurements in-field at the R-6 mine have not been completed and should be considered for future work. A critical convergence velocity of 1 mm/day is recommended for a massive sandstone roof.

Maximum convergence was upto 26 mm in the panel S-1 at the time of first major fall, at time of the about 48 % of panel, maximum convergence recorded was upto 37mm during 4th and 5th major falls, respectively. This increase in convergence can be attributed to cantilever action of the roof layers. And partly to poor conditions in the later in panels regarding roof bolt performance.

On the basis of results of field experimental studies, empirical equations were compare to field value and numerical model at different conditions from the line of extraction during the critical stages of diagonal slicing by continuous miner. The measurements were averaged for different instruments at a particulars distance from goaf edges at the time of pillar extraction. To reduce scatter in the observed data, readings, related to disturbed instruments were not considered in the regression analysis.

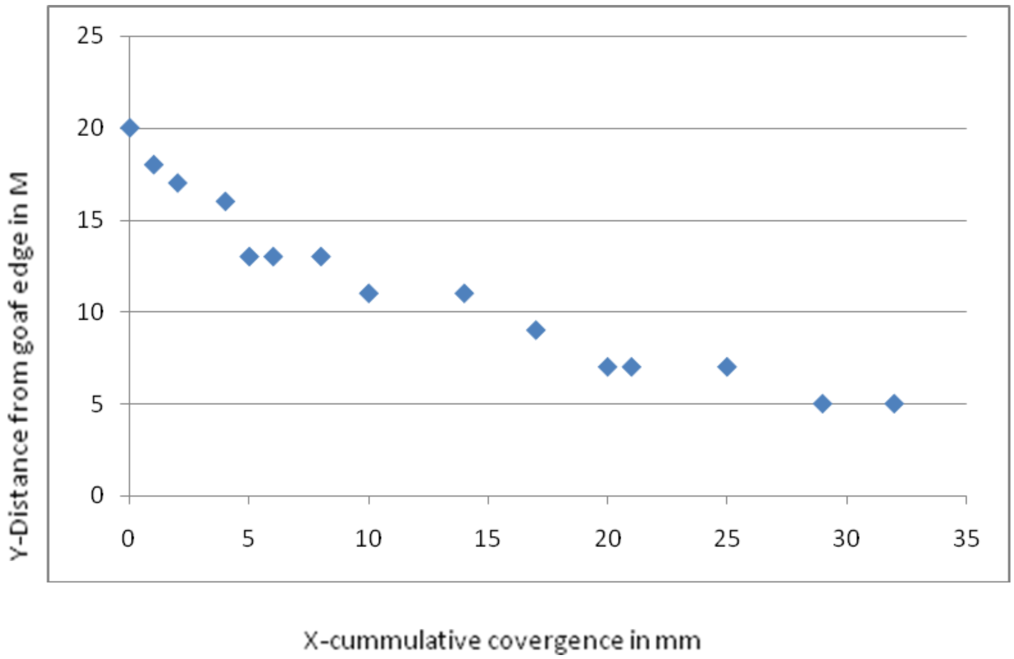


Figure-5.2: Cumulative convergence Vs distance from goaf edge.

Cumulative convergence increased nonlinearly with the distance from extraction line in all the experiments. Its value at different depth covers remained within 26 mm along the goaf line at the time of first major falls, while increased to 55 mm in the goaf. Considering the average values of convergences, the best fit to the regression of convergences (C in mm) and distance from extraction line (D in m) at the time of major roof falls follow the second order polynomial equation with correlation coefficient 0.96.

$$C=23.1 - 1.08\ d- 0.01\ D^2 \quad \text{.......................} (5.5)$$

Results of continuous convergence monitoring data showed interesting phenomena during the major falls. Distinct anomalies were found at 2 hours before the roof fall. Convergence acceleration trend adopted by mandal et al (2004) for Strata behaviour investigation of India's first depillaring face with continuous miner and shuttle car to be more useful for understanding instability rather than simple convergence values on daily basis, which has been a conventional practice in case of 4^{th} major falls in panel, acceleration up to 2 mm / min^2 was prominent two hours before the major fall, which indicate scope of practical application of these results for design of any system for warning of major roof falls.

5.1.2. Support Mechanism

In order to develop a realistic roof behavior model, data presented in chapter 3 and 4 was analyzed in detail. A total 5 junctions and roadway gallery measurement from depth of 75 m to 106 m situated in NCPH colliery, R-6 mine were analyzed in terms of height and magnitude of instabilities in the roof strata. The aim of this analysis was to:

1) Establish at what heights the instabilities took place.
2) How these instabilities can be supported and
3) Establish a roof behavior based on the magnitudes of deformations.

The result obtained from the height of instabilities as per field study in NCPH Colliery , R-6 Mine is limited 2.5M into the roof and there is no evidence of substantial increase in the height of instabilities as is the case in other Coal Mines, (as per literature review). The magnitude of measured deformations is also evaluated against the maximum theoretical deflection in a built in beam using the earlier mention equation-(5.1).

The results obtained from the magnitudes of deformations in intersection and roadway galleries reveal that there is a significant correlation between the underground measurement and the beam theory. Also in the light of the similar correlations found in other sequence of working, it is therefore cancelled that the roof behavior in NCPH Colliery R-6 Mine can be classified as similar to that of a clamped beam.

The results also suggest that based on the height of instabilities measurements and the roof strata behavior correlation shows the suspension sand beam building mechanisms (with improvements as discussed further in this chapter in support mechanism Para) that have been used in NCPH Colliery R-6 Mine as per modified support system, where the appropriate conditions exist. It is however essential to determine/verify the correct support mechanism ensure the stability of roof strata. From the results presented above, the roof behavior model presented in Figure-5.3 is suggested.

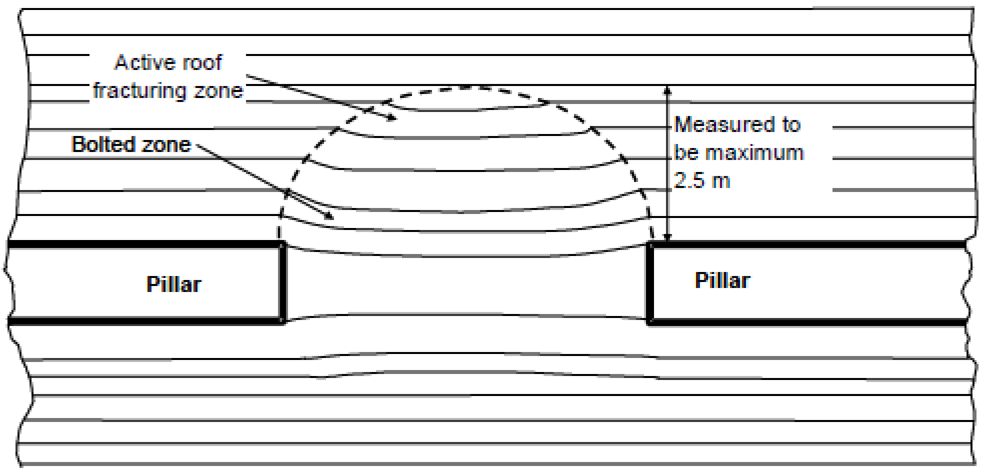


Figure-5.3: Zone of Roof Sagging

This model suggests that when an underground opening is made, the portion of the Strata directly above the opening loses it original support and the stress equilibrium is distributed. The roof strata to sag under the gravitational and / or horizontal forces (Irrespectively) upto a height where there is a competent layer and new equilibrium is reached. In the case of absence of competent layers, as the lower layer start losing their integrity, the height of instabilities increase further into the roof. To maintain the stability, it is essential to keep the immediate sagged zone stable (Fig. 5.3), by using of either suspension or the beam building mechanism. In beam building mechanism, roof bolts in this zone force the entire bolted layer to sag with the same magnitude. The layers within the bolting range thus act like a solid beam supporting the bolted horizon as well as the surcharge load due to sagged layers height into the roof.

As indicated in the above model, before a roof bolt system is designed for a certain support mechanism, it is important to establish the geology for at least 2.5 m into the roof (based on measurements), which will assist in identifying the expected roof behaviour and in determining the support mechanism to be used.

If the immediate roof is very weak, but a competent layer exits higher in the roof, the suspension support mechanism is indicated. However, when the entire roof consists of a succession of thin beams, none of which are self-supporting, the suspension principle cannot be applied in this case beam building mechanism is suggested. It is suggested that before any decision has been made regarding the support system, a detailed geotechnical investigation should be conducted (especially in Greenfield studies) to determine the heights of roof sagging, which can be assumed to be extended up to the “poor” quality layers. This investigation can be carried out using the standard laboratory tests, impact splitting tests, RQD or Rock Mass Rating.

In the suspension mechanism, the lower (loose) layer is suspended from the upper (competent) layer using roof bolts. This creates a surcharge load and increases the maximum tensile stress in the upper layer, above the abutments. This surcharged tensile stress (xx (max) σ in Pa) can be calculated using the following formula;

$$\sigma_{xx(\max)} = \frac{\rho g (t_{com} + t_{lam}) L^2}{2 t_{com}^2} \qquad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ (5.6)$$

Where, ρ = density of suspended strata (kg/m^3)

g = gravitational acceleration (m/s^2)

L = span (bord width or intersectional diagonal width) (m)

t_{com} = competent layer thickness (m)

t_{lam} = laminated lower strata thickness (m)

For stability to take place, the tensile strength of the competent layer should be greater than the tensile stress generated in this layer due to surcharge load. It should be noted that as mentioned above, the thickness of competent layer, the position of competent layer, the bord widths, the thickness of suspended strata and the strength of competent layer will vary in nature. It is therefore suggested in determination of the applicability of the suspension mechanism using Equation (5.6) that a minimum of possibility of stability of (PoS) 91 per cent should be attained.

Regarding the tensile strength of rock mentioned above, it should be noted that the tensile strength of rock is determined by the resistance of rock to tension. The failure of rock under tension is invariably abrupt with total loss of cohesion and load carrying ability. Direct determination of tensile strength for rock, i.e. “pull tests”, is difficult, mainly because of involved specimen preparation. Indirect methods are most commonly used for determining the tensile strength.

The Brazilian (disc) method has proven to be a useful technique for a wide range of rock materials. It has, however, been found that the tensile strength determined by Brazilian tests is usually higher than the direct pull test value.

In general, while a rock material may have a tensile strength, a rock mass is often assumed to have very low tensile strength. This assumption is considered appropriate given the existence of joints and other defects in the rock mass. It is suggested that a detailed analysis should be conducted in determining the tensile strength of coal measure rock.

5.2 Verification of Support System

An extensive literature review revealed that the two support design methods commonly used in Board and pillar by continuous miner for mass exploitation of underground coal deposits.

1. **CMRI-Geotechnical Classification System**
2. **USBM Classification System**

The methods both are divided into rock mass support design methods, rock structure support design methods, and stress support methods. The rock mass support methods include the simple rules of thumb through to rock mass classification methods. All are essentially empirical in nature. The design recommendations from these methods are usually somewhat conservative to account for unknown conditions and anomalous behaviour. This approach is analogous to the application of a factor of safety in the analytical methods.

The rock mass classification methods are the most sophisticated of the empirical design methods. These methods seek to match the support requirements to the quality of the rock mass and site conditions. Rock mass classification quantifies the parameters considered to affect support design thus making it possible to use the methods in a range of geological and environmental conditions. Most of the rock mass classification systems recognize the significance of geologic structure to rock mass behaviour; the input parameters typically include discontinuity properties such as spacing and strength. These methods do not consider discrete blocks or wedges formed by geologic structure. Most of the methods only consider stress in general terms.

The analytical methods match the support requirements to the expected mode of failure. Typical modes of failure include gravity driven rock structure failures and stress driven failures. Prior to selecting a design method, it is necessary to determine the expected mode of failure. Numerical methods are an extension of the analytical methods where numerical simulations are used to analyse mining induced stresses and deformations. Rock failure criteria are then applied to the results of these analyses. Ground support requirements are typically based on the depth and extent of the predicted failure zones around the excavation. Numerical methods are primarily used where failure is being driven by stresses. As failure in shallow mines (R-6 Mine) is seldom due to high stresses, numerical analysis was also considered a useful tool for verifying the support design in this study.

5.2.1 Authentication of existing support system with numerical models

In designing underground supports in coal mines, numerical modeling methods have been widely used to obtain the stress distribution and deformation around entries with which the entry stability are analyzed and desired supports are recommended. The advantage of the numerical method lies in its capability of handling complicated problems and easiness of doing experimental analysis by changing the input parameters. However, the results from numerical modeling are usually not in complete agreement with the field measurements due to the uncertainty of the material properties, geological conditions, and the imperfectness of the model. The gaps between modeling and field measurements can be narrowed down by calibrating the model with field cases.

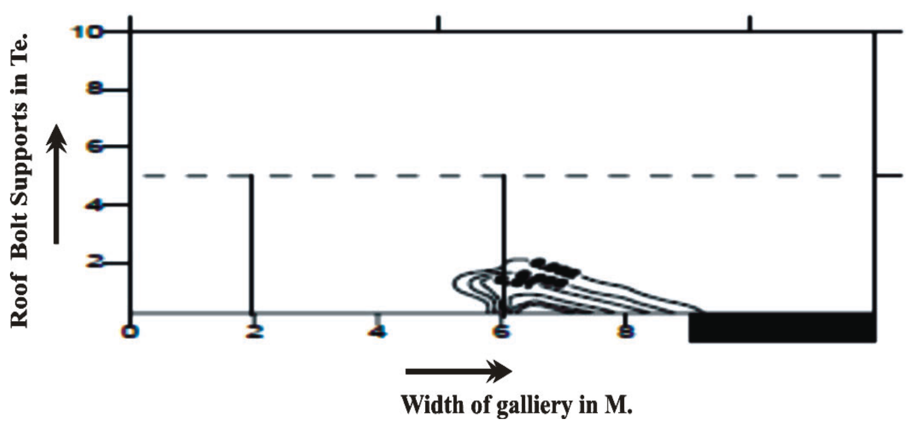


Figure-5.4: Plastic Strain and Failure Modes

Calibration helps adjust the input parameters to the model and make appropriate use of the information the model can provide. Since instrumentation in underground coal mines is costly, good field data are not easy to obtain. Therefore, the data obtained from those rare cases that were well instrumented are invaluable for calibration and verification of numerical models.

In this study, a field case with well-instrumented sites was used to validate the newly-developed 3-D tensioned bolting design model using finite element method. The instrumentation data from an entry intersection were used to compare the difference between the results of modeling and field measurements. Finally, how the model can be used to do the tensioned bolting design is given.

Basic Set-Up for Numerical Modeling and Predictions: Three dimensional numerical models were prepared to evaluate stability of roof rock under various conditions and also to make predictions for continuous miner operations under 6 m wide galleries. All litho logical units with their respective rock mass properties were used for the modeling. Corresponding materials, as per the typical litho logs figure 3.5, have been considered to follow Mohr-Coulomb's elasto-plastic rock failure model with non-associated flow rule. Various rock mass properties and corresponding rock properties are given in Table-5.1.

Table-5.1: Rock Mass Properties used for Mohr-Coulomb Material

Rock Type	Engineering Property	Property of the rock	Property of the rock mass
Coal (RMR=56)	UCS, MPa	28.5	3.16
	Tensile Strength, MPa	3	0.6
	Young's Modulus, GPa	7	1.4
	Poisson's Ratio	0.27	0.27
	Cohesion, MPa	-	0.72
	Friction, Degree	-	41.5^0
Fine Grained Sandstone/ Shaly Sandstone (RMR=47)	UCS, MPa	22.59	1.6
	Tensile Strength, MPa	3	0.44
	Young's Modulus, GPa	4	0.6
	Poisson's Ratio	0.41	0.41
	Cohesion, MPa	-	0.69
	Friction, Degree		35^0

Medium Grained Sandstone (RMR=65)	UCS, MPa	16.33	2.84
	Tensile Strength, MPa	2	0.55
	Young's Modulus, GPa	7	1.96
	Poisson's Ratio	0.31	0.31
	Cohesion, MPa	-	0.7
	Friction, Degree	-	41^0
Coarse Grained Sandstone (RMR=41)	UCS, MPa	9.02	0.47
	Tensile Strength, MPa	1	0.1
	Young's Modulus, GPa	2	0.3
	Poisson's Ratio	0.43	0.43
	Cohesion, MPa	-	0.7
	Friction, Degree	-	37.9^0

The prepared models were provided with gravity loading only as initial load conditions due to the reason that the mine is under shallow depth cover of 87m and there is no sign of distress due to in-situ stresses. Model geometry prepared and used for the modeling. Model boundaries are truncated using advantage of symmetric planes. The positions of symmetric planes are explained in Figure-5.5. Pillar length of 22 m considered to avoid effect of far field boundaries on the simulations. Appropriate roller boundaries are placed at the far field model boundaries. Two categories of models are prepared. The models with 6 m wide gallery were prepared for validation of the modeling while the models with 6.4 m wide gallery were prepared for prediction of roof conditions during continuous miners operations. All simulations have been solved following two stages. Gravitational loads made to settle for initial stage of excavation and thereafter the staged excavation mining steps were incorporated. The models have 38280 zones and 42067 grid points. These zones and grid-points were kept dense surrounding the area of interest, i.e., the mining gallery. Staged excavation with 1m mining step covering 12 mining steps has introduced during the simulations. Models behaviour was evaluated after each simulation through observations of roof rock deformations, material failure state and safety factor contours.

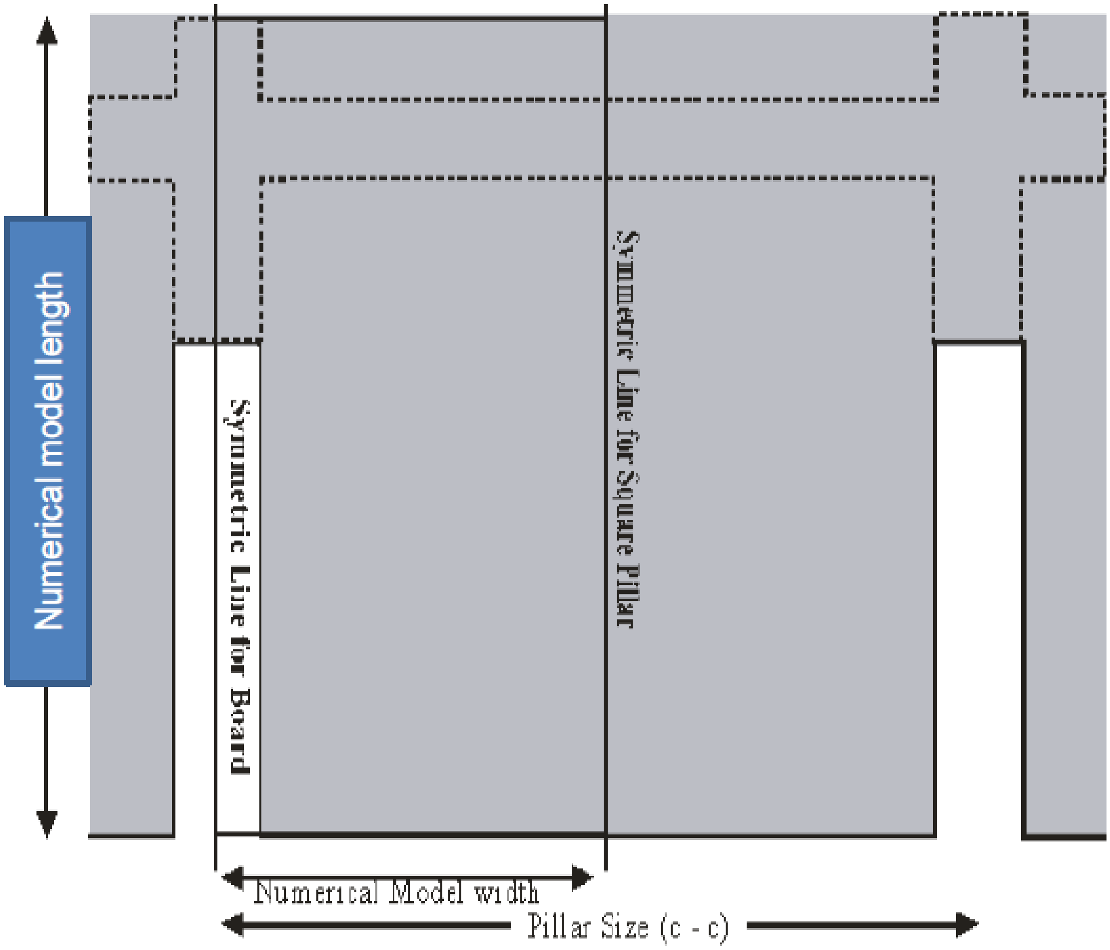


Figure-5.5: Pillar size and symmetric boundary conditions.

Validation of the Modeling: Numerical modeling results are compared with observed deformations values at 3L/16D by multi-pint borehole extensometers (MPBX, observations shown elsewhere) in the mine. Corresponding predicted deformation values through the modelling and the observed deformation values are compared and shown in Figure-5.6. Comparison in the deformation values shows a correlation coefficient of 86% with the slope of the trend line as 25.64^{o}. The high correlation coefficient indicates that the prepared numerical models are accurate enough to provide reasonable trends for the mining conditions.

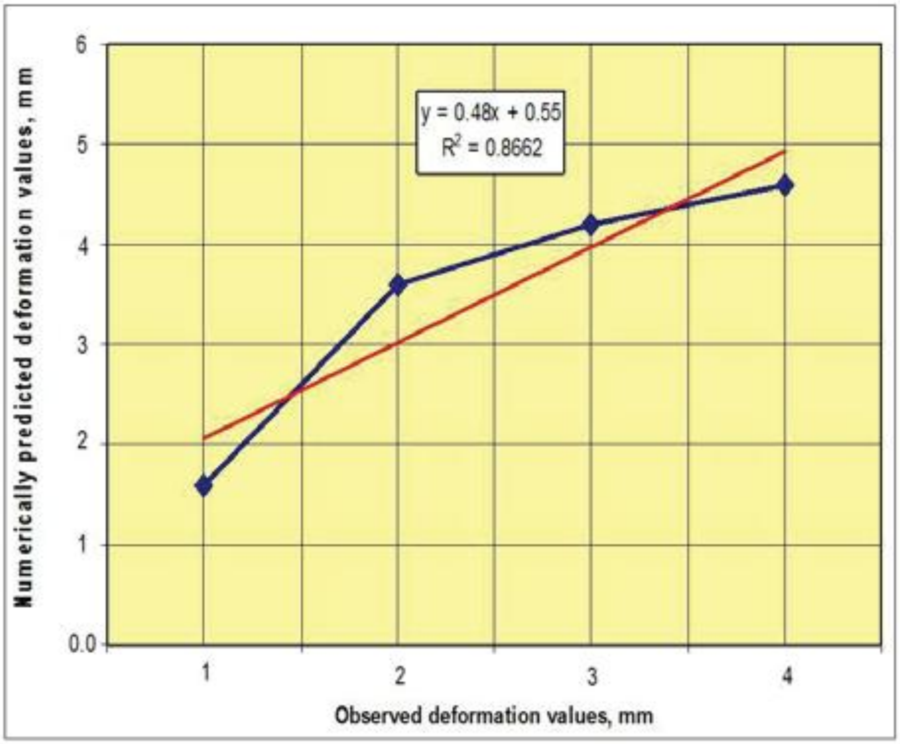


Figure-5.6: Comparison of numerically predicted and observed roof deformations.

Further, no material failure state and corresponding high safety factor contours in the roof strata after 18m of staged excavation simulations (Refer figures no. - 5.7 and 5.8, respectively) corroborate to the prevailing general conditions in the mine. The minimum safety factor contour value is obtained as 2.5 as per the face line while roof level have safety factor values more than 10. These corroborative results are also reinforced later through field observations of deformations through MPBX. The MPBX observations demonstrated deformations in the order of a fraction of 2 mm.

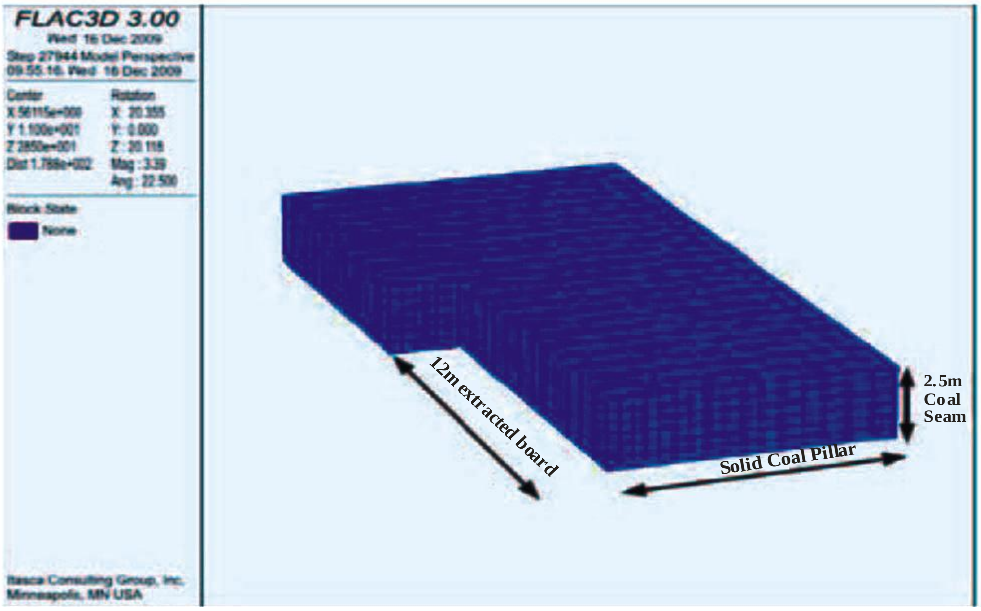


Figure-5.7: Material Failure State in coal seam for 6m wide gallery

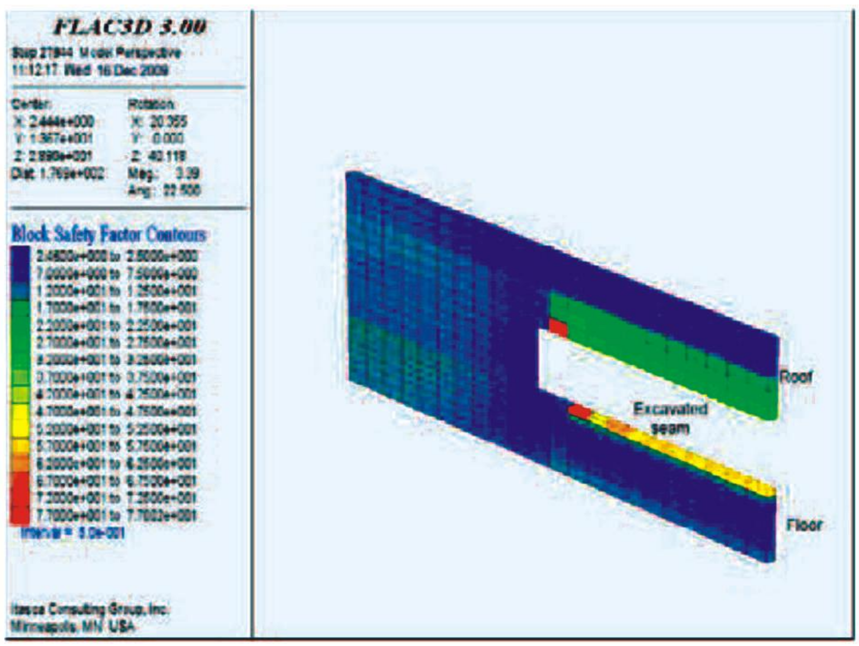


Figure-5.8: Distribution of safety factor contours in roof, face and floor at the centre of bord for 6m.

Predictions of the Numerical Modeling: Material failure state plots (Figure-5.9) and Safety Factor Contours (Figure-5.10) are evaluated to make the predictions for the roof behaviour during the continuous miner operations under 6m wide galleries. The minimum safety factor contour value is of 1.97 at the face while roof level has the safety factor value more than 10. Plots illustrated in Figures-5.10 and 5.11 do not reflect any material change in conditions for the mine from the gallery width widening from 6 m to 6.4 m even after 18 m of staged excavation simulation steps. Further, comparisons of deformation values and support pressure values between corresponding excavations stages of 6m and 6.4m wide galleries indicate that there will be 26% increase in deformation values and no change in support pressure values.

An increase of 26% in deformation means that the deformation values will remain less than 2mm for 6.4 mm wide galleries. This miniscule change in the deformations will not result into any change in support pressure. These observations, like the empirical predictions, predicts that the 6m wide galleries with a cut-out distance limited only by human factor (tendencies to step into unsupported area by CM operator to visualize change in roof strata) and ventilation factor (dust and gas emission) will be safe for the NCPH colliery R-6 mine, M/s. SECL, Bilaspur. Impact of solid pillar size reduction on pillar stability can also be evaluated from a comparative observation of the safety factor contours over coal seam and its pillar formed. Such an observation for respective bord size of 6 m is shown in Figure-5.11. Figure-5.12 indicates that the increase in the bord width does not produce an interaction effect on the pillar size. A reduction in safety factor value from 2.98 to 2.69 with the respective increase in the bords width can only be seen on the side of the gallery up to a skin depth of 0.3 m. The remaining pillar sizes are unaffected and safe under the proposed bord width increase.

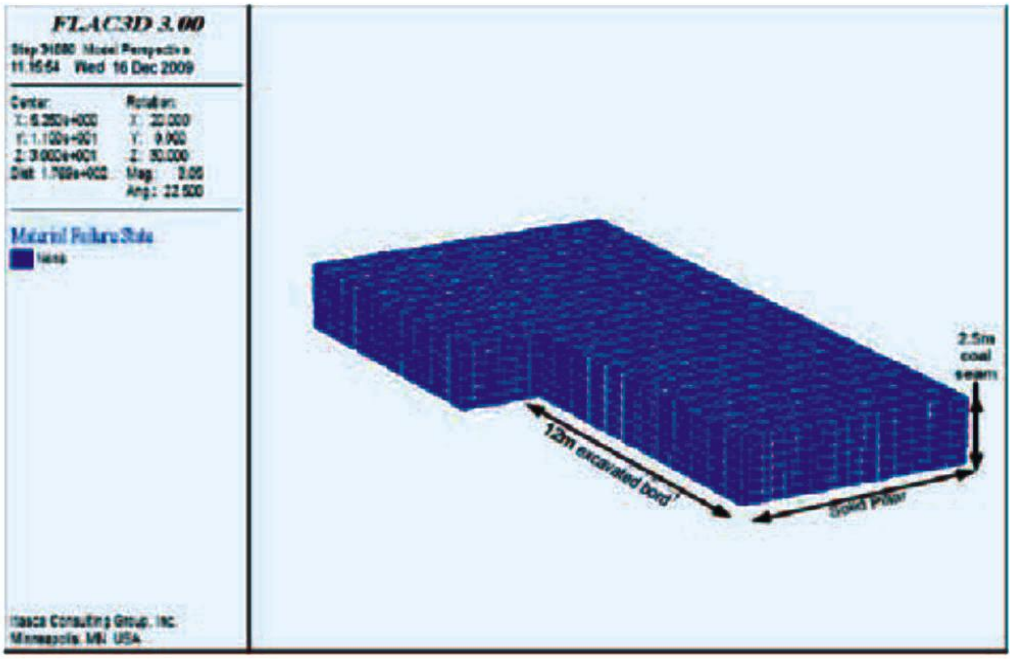


Figure -5.9: Material Failure State in coal seam for 6.4m wide gallery wide gallery

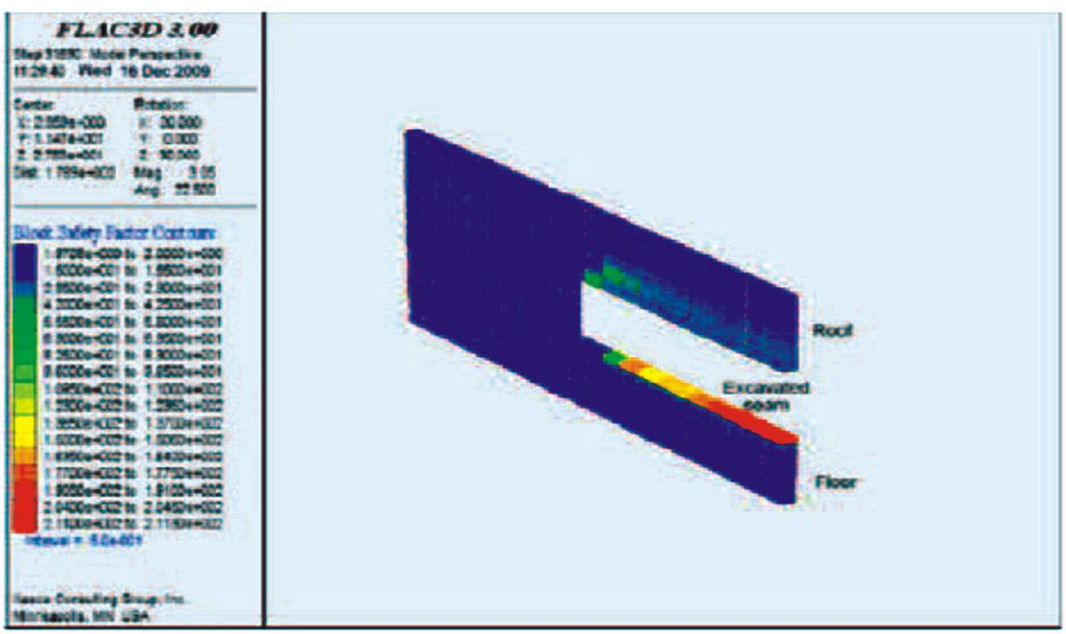


Figure-5.10: Distribution of safety factor contours in roof, face and floor at the centre of bord for 6m wide gallery.

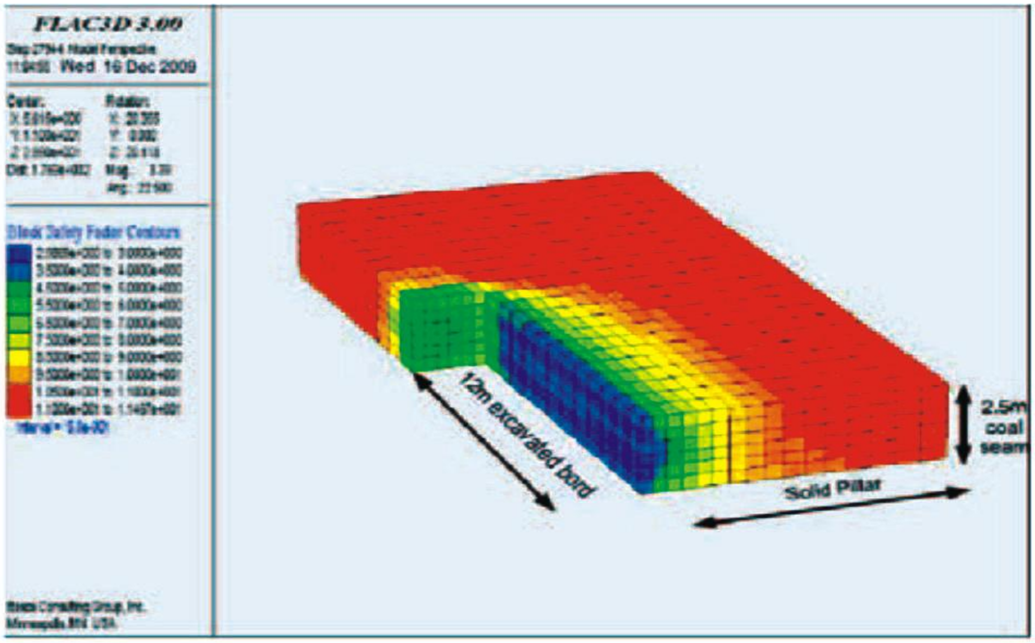


Figure-5.11: Interaction of bord width on pillar size with Safety Factor Contours over coal seam in 6m wide galleries.

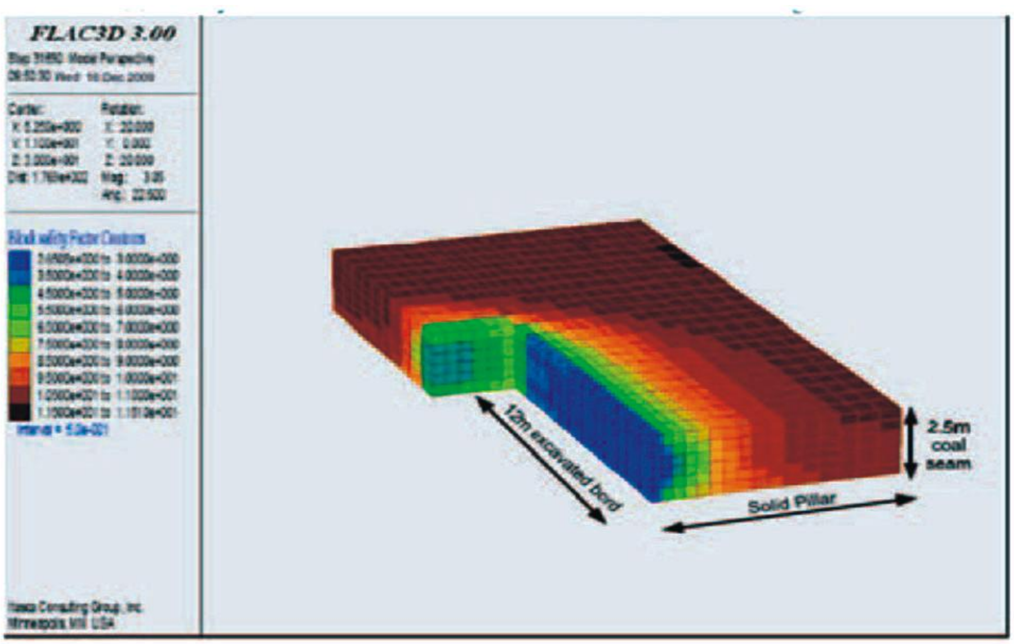


Figure-5.12: Interaction of bord width on pillar size with Safety Factor Contours over coal seam in 6.4m wide galleries.

5.3 Comparison of Results between Modeling and Field Measurements

5.3.1 Roof Bolt Loading

The bolt load changes were monitored as the intersection was widened for a period of time. After widening of gallery junction the measured roof bolt load changes available shortly or within 24 days, (Hanna, et al, 1991). Notice that the bolt loads concentrated more on one corner of the intersection due to the effect of the slips. Since the corner around the pillar A was farthest from, and subjected to the least influence by, the slips, the sub model, which consisted of a quadrant of the intersection, was chosen from this area to obtain the bolt load changes. Figure-5.13 shows the load changes for the bolts around pillar A from both the modeling and field measurements 24 days after the intersection widening. The bolt locations in the model are not exactly the same as those in the field, but the numbers of bolts installed are the same. The measurements showed that the bolt loads at the center of the intersection were larger and those around the entry were much larger than those around the crosscut.

The larger bolt loads around the entry area were attributable to both the existence of slips and major horizontal stress. The predicted load by the model is also the highest around the center of the intersection and larger around the entry than around the crosscut because the entry direction is almost perpendicular to the major horizontal stress. The predicted loads by the model are close to those by field measurements around the center of the intersection and crosscut but lower around the entry. This difference could be caused by the slip near the pillar and the slip across the pillar A.

5.3.2 Roof Displacement and Bed Separation

Roof displacement and bed separation were monitored by extensometers. Analysis of the data was based on the assumption that the top anchor remains in a fixed position over time. Movements of the other anchors, including the roof surface, were calculated relative to the position of the top anchor. At each station, the data were used to calculate the downward movement of each anchor and the change in distance between anchors. The change in distance between anchors corresponds to the opening and closing of bed separations that occur between the anchors. Figure-5.14 shows the extensometer locations around the intersection.

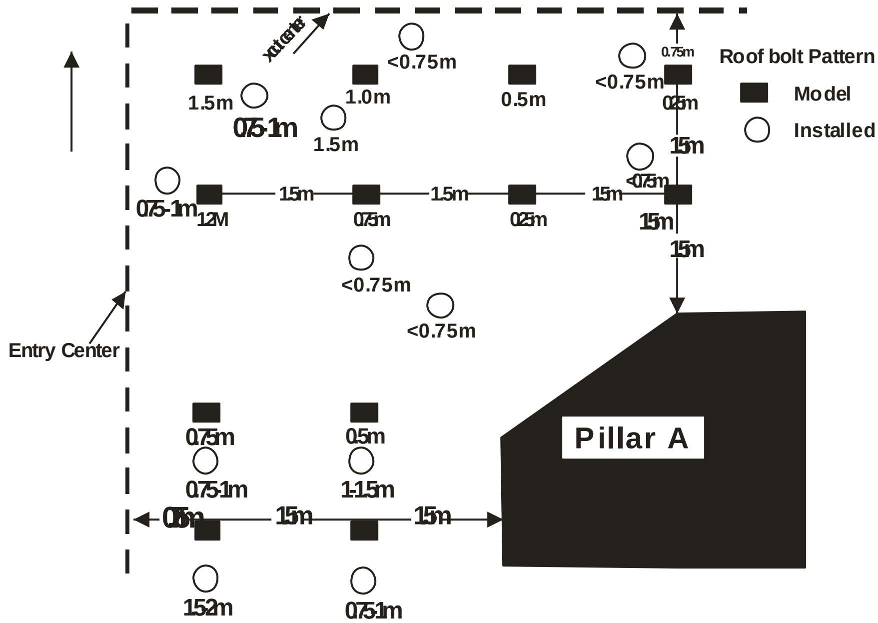


Figure-5.13: Measured and Predicted Changes in Bolt Load after junction after widening

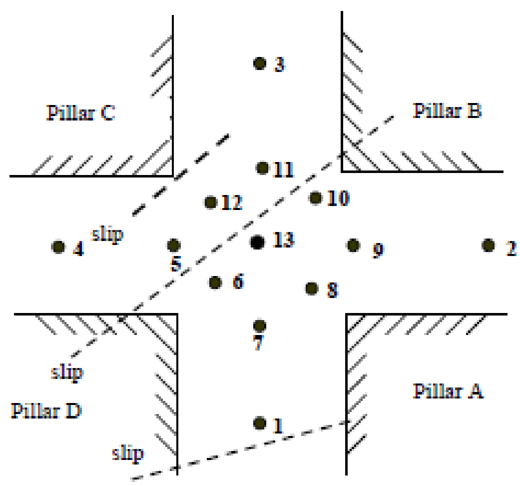


Figure-5.14: Extensometer Locations

Figure-5.15 shows the displacement of the roof line at the locations shown in Figure-5.13 from the modeling and field measurements 6 days after the widening. It can be seen that the measured roof line displacements were close to those from the modeling except at extensometer Nos. 1, 5, 11 and 12 that were close to the slips. It can be concluded that the model can predict roof displacement fairly well when the effects of geological anomalies such as slips are excluded.

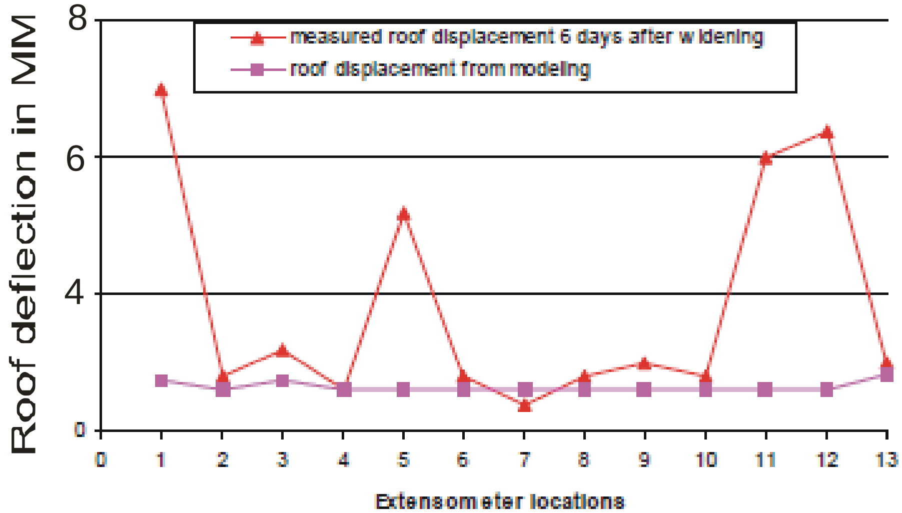


Figure-5.15: Roof Line Displacement

Figure-5.16 shows the downward movements of each anchor for three extensometer stations at points 7, 8 and 9 shown in Figure-5.14, which are away from the influence of the slips. The four anchor locations are at the depth of 0, 0.5m, 1.0m and 1.5m, respectively. The extensometer data in Figure-5.15 were chosen from 6 days after widening when cut sequence 6 m wide and 35m advancement was complete. It is shown that there was no separation between 0.5m and 1.0m at extensometer 7 and 8 and that there was a differential displacement of 8mm at extensometer 9 between 0.5m and 1.5m, which could be caused either by plastic deformation or bed separation. Therefore it may be concluded that no separations or only extremely small separations exist between roof surface and 1.0m horizon, especially at the interface of sandstone and shale at 0.75m horizon.

The results from the modeling shows that there was a separation along the bedding plane 0.75m above the roofline before roof bolts were installed (Figure-5.15). The separation at the center of the intersection was about twice as much as that at the center of the entry. But the separation was closed after installation of bolts with 9-ton pretension as shown in Figure-5.16. Therefore, the model can predict closure of separation in the immediate roof resulting from bolt installation.

5.3.3 Roof Yielding

The extensometer data at the locations 7, 8 and 9 show that there were relatively large displacements six days after widening between roof surface and 0.5m horizon while the displacements at 1.0m horizon were zero or very small (see Figure-5.14), which showed the depth of roof yielding was about 0.75m. The results from the model also show that the yielding zone is within 0.75m from the roofline. Therefore, the yielding zone obtained from the model could be used to estimate the roof yielding in the field.

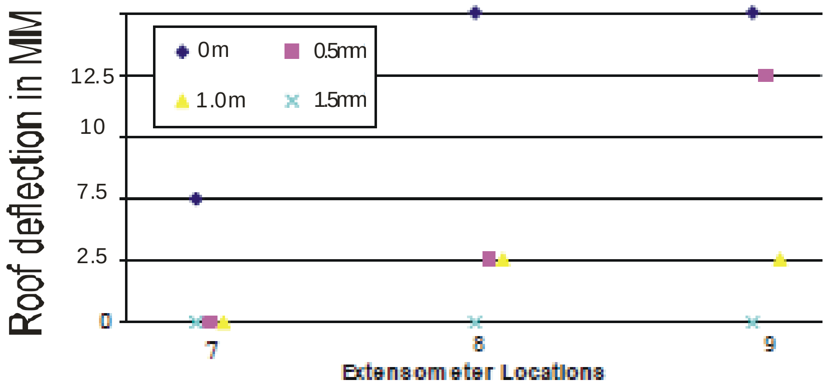


Figure-5.16: Measured Roof Displacements at Different Horizons

5.3.4 Tensioned Bolting Design

This case study has demonstrated that the modeling results matches fairly well with the field measurements shortly after the intersection widening, from which the procedure for the design of tensioned roof bolting can be developed as follows: First of all, the bolt length can be determined by the depth of the yielding zone obtained from the model. The designed bolt length is the depth of the yielding zone plus the anchor length.

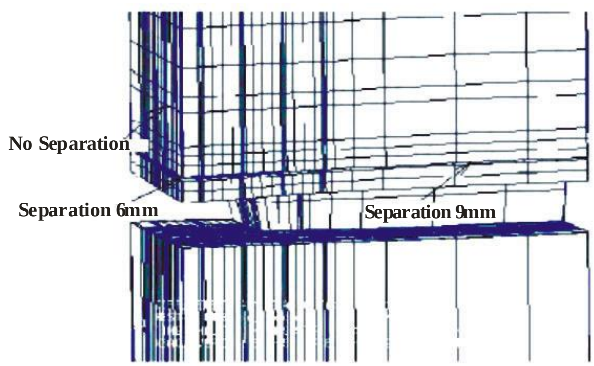


Figure-5.17: Bedding Plane Separation before Bolt Installation

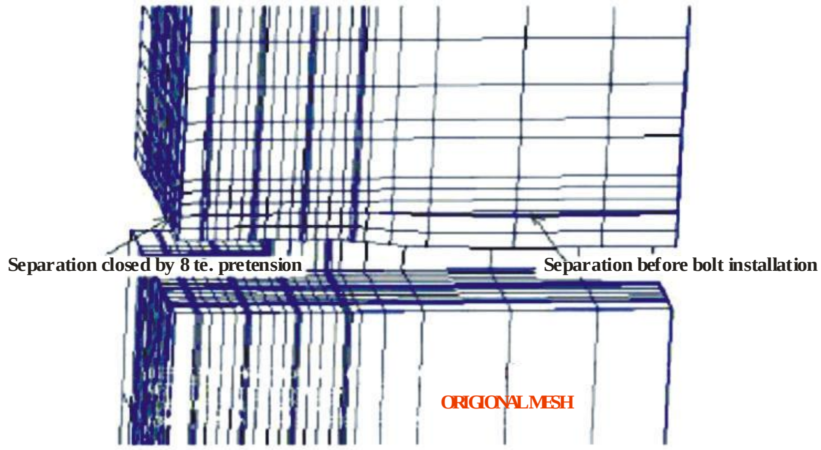


Figure-5.18: Closure of Bedding Plane Separation by Tensioned Bolts

Secondly, the optimum pre-tension is another important parameter for tensioned bolting design which can be obtained from the model. In past various study also showed that the tensioned bolts are to reinforce 0.75m-1.0m of the immediate roof and suspend the bolted strata to the upper stable roof and therefore the beam building effect of the tensioned bolts is within 0.75m-1.0m of the immediate roof. Consequently, the optimum pre-tension required should be that which will close the separations within 0.75m-1.0m of, and provide a compressive zone, in the immediate roof. To obtain the optimum pre-tension, the model can be run with incremental pre-tension until the separations in the immediate roof are closed. The tensioned bolts are effective for the layered immediate roof. If there is no bedding plane in the immediate roof, the tensioned bolts should provide a compressive zone, about 0.75m deep, in the immediate roof.

Thirdly, the load in the bolts obtained from the model can also be used for the tensioned bolting design. The load in the bolts is the combined result of the pre-tension and roof deformation, which is related to roof strength and horizontal stress level. Since the tensioned bolts are designed to work within its capacity, the designed bolt load should not exceed 75% of its capacity. If the designed bolt load is beyond this limit, the bolt density should be increased to reduce the load in the bolts.

In this case study, the yielding zone in the roof was 0.75m deep, a 9-ton pre-tension could close a separation in the immediate roof and the loads in the bolts from the model are not too large. So the roof bolting design was appropriate based on the modeling.

From the field measurements, the intersection supported with 1.5 m tensioned bolts with a 9-ton pre-tension were stable except a small roof fall near a slip.

5.4 Guidelines for Strata Management

India has large resources of coal deposits for underground mining and lot of coal was blocked in existing underground mines. Safe extraction of these can be made possible by effective strata management. Accidents due to movement of strata in underground coal mines had been a major concern for the mining industry and it is largest contributing factor of underground coal mine accidents. Continuous efforts were being made by all concerned to reduce the hazard of strata movement. Manager of every underground coal mine shall prepare a strata management plan basing on RMR, RQD, Q-classification, structural mapping, geological disturbance, geo-mechanical parameters, design of support, SSR etc. and copies of the same shall be distributed to concerned officers and the plan also shall be updated time to time as per the progress of workings and whenever additions and deletion in the instrumentation.

The strata evaluating officers, strata instrumentation officers, strata monitoring officers, geologists, scientific assistants shall be regularly trained and retrained in the strata behaviour management at reputed scientific agencies in India and abroad to update their knowledge in the relevant field. Standard for the development of strata management plans

Purpose: The purpose of this guideline is to provide underground coal mining operations with a structure for the development of site specific Strata Management Plans (SMPs)

The purpose of an SMP is to

a. Ensure a risk based approach is taken;
b. Ensure that introduced risk controls are monitored to ensure their effectiveness
c. Ensure the regular review of the strata management plan

Objective: The objective of a Strata Management Plan (SMP) is to ensure the following conditions are satisfied:-

a) Appropriate levels of support are designed, installed, maintained, and monitored throughout all stages of underground coal mining operations.
b) Site-specific “Support systems” for each panel or development heading and pillar extraction are implemented.
c) Obligations under the Coal Mines Regulations 1957 and the DGMS technical circular are met or exceeded.

Routine monitoring and mapping of roadway and pillar conditions and the timely assessment of additional strata control measures are determine and acted upon. Likewise, monitoring, mapping and formal technical, assessments are to be the basis for reducing or increasing strata control measures or for introducing new support systems.

- The SMP is to provide a “living,” technical and operational based decision-making process by which the ground support needs of any given roadway can be assessed.
- All technical assessments are to mitigate risk.

5.4.1 Relationship of the Strata Management Plan: Strata Management Plan is just one of a quantity of management plans that should be used an underground coal mining operations. The framework for all site SMPs shall include but is not limited to:

- The “documents” or the SMP itself and related appendices and safety management plans;
- A documented Technical Risk Assessment that includes a review of all Strata Control issues and processes;
- Process of routine strata control assessment (hazard mapping, monitoring);
- Monitoring of strata movement around roadways (databases), interpretation of the data and as required technical review;
- Installation of the required support and monitoring equipment (Manager’s Support Rules and Support Installation Sequence Plan);
- Process of review, auditing and authorization;
- Communication of the plan, and;
- Training.

The Site Specific Strata Management Plan Document: Strata Management Plan should contain the following key elements:-

- Introduction/Objectives
- SMP Scope and excavation method
- Definitions if new terms are used
- Hazard Identification and risk assessment
    - Geotechnical/Geological Factors
    - Operational Factors
    - Technical Risk Assessment (TRA)
- Control Process
    - Standing order of Managers Support Rules
    - Support Installation Sequence Plan
- Design Process
    - Development Support System
    - Design of Excavation Support System – peer review
    - Fall of ground/exceptional circumstances
- Resources and their Roles and Responsibilities
    - Strata Management Requirements
    - Workforce (including contractors)
    - Other Key Strata Management necessities
- Communication
- Training
- Installation and Inspections
    - Installation
    - Inspections
- Monitoring
- Excavation planning and Hazard Map
- Document Control

- **Introduction/Objectives:** This section of a site SMP will briefly describe the excavation to be undertaken and principle strata control hazards.
    - Geothermal characteristics
    - Depth of Excavation
    - Excavation Method
    - Shift work system and management structure;
    - Ventilation
    - Access to/from the heading/face;
    - Current and future underground workings design(s)

The objectives shall include, but not be limited to:

- Appropriate levels of support safety are designed and installed.
- All excavation supports are maintained and monitored throughout all stages of the excavation.
- Hazard map is reviewed and results monitored.
- Routine monitoring of roadway conditions and the timely assessment and implementation of additional or altered strata control measures.
- Providing a technical and operational based decision-making process by which the ground support needs of any given roadway or tunnel can be assessed.
- Develop a “living process” to meet the above objectives that involves all levels of the workforce.
- **SMP Scope and Mining Method:** The scope shall state that the SMP relates to underground excavations design, implementation, monitoring, and performance review, and will cover, but not be limited to:
    - All types of roof and rib support;
    - Inspection of roadway conditions;
    - Monitoring of strata movement around roadways and voids;
    - Installation of required support;
    - Key individual responsibilities.
    - Communication of the plan
    - Training
    - Auditing and peer review.

An SMP does not cover ventilation, inrush, or subsidence; however, an SMP is likely to have a close relationship with these, and other, management plans.

- **Hazard Identification –Risk Assessment:** A formal, documented technical risk assessment (TRA) shall be performed for strata and geological hazards for all excavations prior to development of its SMP. This does not preclude the need for strata related TRAs if changes in ground conditions fall outside those originally assessed in the TRA.

  The TRA should evaluate hazards, assess their probability and consequence, and determine risk mitigation measures.

The risk associated with strata and geological related hazards should be estimated by considering the consequence and probability of the hazard. In some circumstances, the probability of a potential failure may be quantified from past ground failures. A Register shall be used to record all failures that occur in supported or unsupported ground. These are to be treated as an accident and reported and investigated.

It is recommended that the strata TRA be done as part of the SMP. All technical risk assessments shall be documented and retained on file.

- **Geotechnical/Geological Factors:** Factors that shall be considered but are not limited to:
    - Adequacy of exploration data (stratigraphy, lithology, geophysics, rock mass and defect strength)
    - Adequacy of In Seam/mine Data (quality, seam thickness variations for mines)
    - Adequacy of Interpretation of Data
    - Regional variations in the Stress Field
    - Structural anomalies identified
    - Roadway and tunnel design model adequate for development and extraction and for life of usage or project
    - Pillar strength modeling methodology to industry best practice
    - Permanent and temporary lining design modeled for all conditions, design life and usage

- **Operational Factors:** Factors that shall be considered include but are not limited to:-
  - Management of stress concentrations
    a. In underground excavations
    b. During extraction
    c. Prior to first major void fall and all on-going void falls
    d. Post extraction (multiple seams/veins)
  - Adequacy of Manager's Support Rules covering, installation, availability and suitability of support hardware for use and storage
  - Identification of changes in conditions and mitigation responses
  - Understanding of risk and consequences of failure
  - Pillar and Roadway dimensions – must be geotechnically appropriate
  - Management of roof and/or rib falls, pillar crush, guttering, roof squeeze and/or floor heave, deformation and/or failure of support mechanisms.
  - Slurry TBM, slurry and circulation
  - Void filling behind installed lining systems as required.
- **Control Process:** The Mine Manager has ultimate responsibility for ensuring that the SMP is complied with. Under the SMP certain responsibilities may be delegated to people in appropriate roles that are required for the development, implementation, and continuous improvement of the plan.
- **Standing Order of SMP, Manager's Support Rules, and Hazard map:** The Mine Manager shall have sole authority to approve and issue the Manager's Support Rules, Support Installation Sequence Plan, and the SMP. Temporary changes or deviations from the Manager's Support Rules will require a formal risk assessment and the approval of the Mine or Tunnel Manager, and may require the review of a Geotechnical Engineer or Senior Mining Engineer. Any changes to these documents, shall be recorded, filed, and distributed to all concerned.

A copy of the SMP, all Hazard Map, and Manager's Support Rules shall be kept in the under manager, and Mine Manager's offices. In mines the under manager shall ensure that all Production Supervisors responsible for roadway and pillar stability, during development and extraction, are familiar with the content and their responsibilities under the SMP, Hazard Maps.

In mines the under manager shall ensure that a copy of all Hazard Maps is available in all crew crib rooms or on panel notice boards.

- **Design Process:** All documentation plans and logs relevant to the design process shall be kept and filed for the duration of the mining operations. Prior to commencement of underground mining associated with development or excavation, all information available pertinent to the support system design shall be assessed. If the information is insufficient, recommendations for further work to address the gaps shall be made and completed. No roadway development, tunneling or extraction should take place in an area that has not been assessed and a support system appropriate to the prevailing conditions established and recorded. The geotechnical assessment should include, but is not limited to the following:

  a. Roadway and/or pillar dimensions
  b. Life and uses of roadway/excavation
  c. Method of extraction/development
  d. Location of cross cuts and stubs
  e. Fall or caving characteristics
  f. Performance of similar roadways/excavations
  g. Monitoring results
  h. Structural geology interpretation and extrapolation between panels
  i. Seam/vein thickness and dip
  j. In-situ stress regime (direction and magnitude)
  k. Excavation depth
  l. Physical inspection (roof/floor/rib/void/lining conditions)
  m. Gas drainage data
  n. Groundwater
  o. Qualitative and quantitative assessments of current and previous strata support systems
  p. Exploration data
  q. Back analysis of failures
  r. Roof, rib and floor lithology and rock properties

s. Performance history of support and lining methods in previous sections or mines or tunnels within the region (e.g. Percentage of gloved bolts, encapsulation tests, load transfer data)

t. Available modelling data.

u. Specifics e.g. monitor or continuous miner, lining system, pipe, and conveyor installations.

- **Development of Support System:** Following assessment of the relevant information, a support system appropriate to the expected conditions shall be designed. Consideration of the following (but not limited to) should be given:

  I. Type of mineral deposit and its depth

  II. Geology and geological discontinuities

  III. Physical property of the strata

  IV. Mining method of extraction

A support system may include, but not be limited to:

- Primary support
- Roadway
- Supplementary support (suitable for long term use of roadways and including extraction requirements)
- Monitoring equipment
- Pillar dimensions/reinforcements
- Method of development
  a. Timing of support installation
  b. Sequence of driveage development
  c. Seam thickness, roof and floor coal thickness
- Concrete lining – segmented, pipe jack, insitu concrete

- **Extraction Support System for Underground Excavations:** Prior to extraction and following assessment of all relevant information, a support system appropriate to the prevailing conditions shall be designed. This may include but is not limited to:

  a. Supplementary support (i.e. support in addition to that installed on development)

  b. Intersection support

c. Consolidation of bad ground or faults e.g., structures/zones of weakness.
d. Passive support (e.g. Props)
e. Monitoring
f. Work procedures for extraction

5.4.2 Resources and their Roles and Responsibilities

Strata Management Team or named engineer shall be responsible for ensuring that the site strata management plan is developed, implemented, reviewed, and revised. Ultimate responsibility lies with the Mine or under Manager. The site will determine the appropriate levels of support, monitoring, and hazard response for all underground excavations. Where appropriate, monthly strata management team meeting shall be held and documented.

- ❖ **Strata Management Responsibilities:** Strata Management needs to be controlled by the Mine or under Manager. The strata support design should be completed by a suitably qualified professional engineer taking into account input from a geologist or geotechnical engineer. The strata designs should be peer reviewed before used on site. On site strata support reviews should be carried out to take into account changing ground conditions. The review may be carried out by Strata Management Team consisting of-.
    a. Mine Manager
    b. Geotechnical Engineer/Engineering Geologist
    c. Under manager (as per CMR1957)
    d. Geotechnical Consultant (as needed)

Mine or under Manager Shall –

- o Oversee and drive the Strata Management Plan and ensure the SMP is audited annually.
- o Appoint and ensure that the necessary resources are provided to manage the SMP.
- o Ensure all persons working underground undertake appropriate training in Strata Control.
- o Ensure the mine or tunnel complies with all requirements of the SMP.

- **Strata Management Requirements:** Strata problems include roof sagging, floor heaving, guttering in the centre of galleries, side spalling, presence of shale or clay bands, heavy water seepage, faults, cleavage planes, cracks, joints etc. State of art of monitoring system through instrumented rock bolts, tell-tales, multiple point bore hole extensometers, convergence indicators, local cells etc. are available for continuous monitoring the strata movement. Strata control instruments are helpful in analyzing the deterioration in the roof and indicates roof movement. The monitoring of the effectiveness of roof bolts (primary type of supports) in the active working areas helps in taking critical decisions like modifications of SSR, withdrawal of work permits in the event of any danger from strata movement.

  In line with the 10th National Conference on Safety in mines held on 26th & 27th November, 2007 at New Delhi recommendation and DGMS technical circular guidelines, every Mining Company already established a Strata Management Cell and for its effective functioning the following manpower and instruments are required:-

  - **Man-Power:** As per guidelines mentioned in DGMS technical circular
    - Corporate Level
    - Area Level
    - Mine Level
  - **Instruments:** Minimum numbers of instruments to be made available in working order and maintained at respective levels are shown in Table-5.2. These instruments are in addition to the actual instrumentation already commissioned in the mine as per the local requirement.

Table-5.2: Minimum number of instruments to be made available at respective levels

Sl. No.	Type of Instruments	Corporate	Area	Mine
1.	Tell-tale (Mech.)	2nos	20nos	4nos
2	Multi-point Bore Hole Extensometer	2nos	20nos	2nos
3	Tell-tale (Electronics)	2nos	20nos	4nos
4	Spring loaded convergence recorder	2nos	50nos	2nos
5	Telescopic Convergence Recorder	2nos	20nos	2nos

6	Remote Convergence Indicator	2nos	10nos	2nos
7	Stress Meter	2nos	30nos	4nos
8	Mechanical type/hydraulic type load cell	2nos	30nos	6nos
9	Electronic load cell	4nos (2*+2#)	20nos (10*+10#)	2nos
10	Flat jack	2nos	10nos	1no
11	Earth pressure cell	2nos	20nos	1no
12	Crack Meter/Joint Meter	2sets	10sets	1no
13	Strain Gauge Rock Bolt/Instrumented Bolts	2nos	20nos	2nos
14	Magnasonic Extensometer	1no.	2nos	1no
15	Chart type convergence recorder	2nos	2nos	1no
16	Tape extensometer	2nos	2nos	1no
17	Electronic distance measuring tape/other measuring tape	1+4nos	1+4nos	1+2nos
18	Data logger – with related computer	1set	1set	1set
19	Read out unit of each and every electronic – 1 set each equipments & installation tools	2sets	4sets	1set
20	Strata Control Software packages	2nos	2nos	1no
21	Digi cam	1no	1no	-
22	Lap top	2nos	2nos	1no
23	Borehole TV Camera	1no	1no	-
24	LCD Projector	1no	1no	-
25	Ground penetrating reader (GPR)	1no	1no	-

* Prop type

\# Roof bolt type

Instrumentation Plan in Bord & Pillar Method by Continuous Miner operation

1) Tell-tale & convergence recorder at every junction
2) At least 5 strain gauge rock bolts, 5 magnasonic extensometers in the central part of the panel.
3) At least combination of 5 RCIs & 5 stress meters in the central part of the panel in every alternate pillar.

- **Other Key Strata Management necessities are-**
  - Any areas that exceed design dimensions must be reported to the mine manager.
  - Accurate and up to date plans and records of the underground workings shall be maintained.
  - A comprehensive training and assessment plan shall be developed and implemented. Records of any training and assessment conducted shall be maintained.
  - An adequate and useable, supply of support and strata control materials shall be maintained.

5.4.3 Communication

Clear communication process shall be implemented. The process shall ensure that:-

a. Operators/underground workers are provided with an understanding of expected conditions, anticipated support, underground mining procedures, and any relevant changes in support design prior to implementation.
b. Personnel are aware of typical warning signs, which suggest that the installed support may be inadequate, and needs review.
c. Close communication exists between all members working under the SMP. This includes communicating effectiveness of SMP to site management.
d. Site management has an early opportunity to respond to unexpected excavation conditions and/or support system behaviour.

Communication channels may include but are not limited to-:

a. Hazard Maps and cross-sections
b. Support rules and drawings
c. Strata Management Team meetings and minutes

d. Shift reports
e. Relevant monitoring systems
f. Audit reports
g. Toolbox talks
h. Hazard/accident reports

5.4.4 Training

The Mine Manager shall develop a plan, which shall include the following:

a. Assessment procedures and schedules for determining the competency of personnel to all aspects of strata control (support planning, installation, monitoring, strata hazard identification etc) and management standards and requirements.
b. Identify and record personnel requirements for training and training undertaken of the above.

5.4.5 Installation and Inspections

Support installation and strata control inspections shall be carried out to prescribed and documented standards in the SMP.

❖ **Installation:** The designed support shall be installed in accordance with the established standards and procedures, Support Installation Plan, and manufacturers recommended methods of installation. All underground workers responsible for installation of support shall be formally trained and assessed by the Manager or a suitably qualified person approved by the Manager. Personnel should not carry out roof support work under an unsupported roof. Shift supervisors shall:

a. Observe the ground conditions and installation effectiveness,
b. Report any adverse conditions to the person responsible for Strata Support.

If roadway support consumables or equipment are defective, such as to compromise the ability to adequately support the roadway or tunnel, it shall be repaired or replaced before further excavation is undertaken or support is installed. Sites should ensure that when handheld bolters are used the air pressure is running at 95psi or greater, or as per manufacturers specifications. The shift supervisor should ensure that:

a. The support mechanisms installed in a shift are undertaken by competent staff
b. All drill bits used comply with the requirements defined in the Strata Support Plan or supplier recommendations,
c. Compressed air for handheld bolters is clean and dry and that a "turtle" or oil bottle is installed in the air supply line as required,
d. Air pressure when the bolter is operating shall be at recommended pressure.

A maximum cut out distance shall be defined in the Strata Support Plan and will cover the following:

a. Development or tunnel headings
b. Secondary (e.g. sublevel or cross-cut roadways) development roadways
c. Special infrastructure sites (e.g. screen bays, pump stations, belt change overs)
d. Extraction (i.e. CM or monitor)

- **Inspections:** All underground excavations shall have regular and standardized inspection procedures. The Coal Mines Regulations 1957 specifies required minimum standards for inspection in underground mines. Failure to maintain approved and correct inspections and reporting procedures for strata control can cause the following situations:
  a. Inadequate measurement and monitoring of the underground environment such that accidents may occur which risk affecting the health and safety of personnel and the underground mine.
  b. Inadequate inspections and reporting such that statutory obligations are not complied with.
  c. Inadequate reporting to meet company requirements.
- **Recording Requirements:** An 'Operations Record' as defined in the Coal Mines Regulations 1957, Regulation 108, shall be kept at each underground organisation office. This record shall contain the following information:
  - Date, time, and findings of every examination, which shall be carried out as follows:
    - Before the start of each shift, and during each shift, every area of the underground organisation where personnel are or will be present; and
    - At least weekly, every other accessible part of the underground organisation; and

- **Face and Working Place Inspection:** The Manager or delegate shall ensure the face; roof, ribs, and extraction areas are inspected and assessed for hazardous strata conditions, stability, and support and underground worker safety. This inspection shall be carried out and recorded within two hours of an oncoming shift and at regular intervals during the shift, but not more than 8 hours apart. The timing of this inspection corresponds to the Ventilation Inspection requirements and therefore could be performed together.

  All underground personnel shall report an unplanned fall of ground to the responsible Shift Supervisor or the Manager who in turn shall document it in a hazard/incident register and ensure that the Manager is informed of the fall. The Manager or delegate shall then inspect the area, and compile a report on the key geotechnical contributing factors and strata management plan compliance and shall ensure the fall is fully reported.
- **All Other Accessible Areas of the Mine Inspections (for mines only):**The Mine Manager shall establish a plan for mine officials to carry out the following inspections for the general condition (this includes Strata Control) of the mine at least once per week:
  a. Main roadway returns airways
  b. Each air district returns airways
  c. All seals and ventilation structures
  d. All accessible falls and area.
  e. All general condition inspections shall be documented.

5.4.6 Monitoring

The Geotechnical or Design Engineer shall ensure a formalised monitoring programme, that determines the support effectiveness and ground behaviour is established and documented.

Monitoring should include:

a. Visual inspections
b. All installed monitoring equipment
c. Air pressure at the face or support installation area for effective air supply during installation

d. Pre-tension applied by all pre-tensioning equipment shall be monitored on a systematic basis by the operator. The pre-tension will be audited and the results documented every 6 months to ensure that the support system is being installed to its designed capability.

As a minimum, the following parameters require quantifying, monitoring, recording, and interpreting:

a. Virgin or stress field environment
b. Mining induced stresses from extraction
c. Long term excavation and pillar stability (assessment of actual vs. planned/modeled)
d. The Monitoring Plan shall clearly state the schedule and personnel responsible for the various monitoring required in the plan.

5.4.7 Mine Mapping and Hazard maps

The Mine Manager or delegate shall ensure all roadways and excavation covered by the SMP are mapped and interpreted within 2 weeks of it having been excavated. Mapping should aim to identify all hazards related to strata conditions at the time of mapping. Retrospective or periodic back-bye mapping shall also be performed and stored.

The mapping shall be recorded and interpreted by a suitably qualified person all paper and electronic copies shall be kept in perpetuity as required under the code of practice. The hazard mapping interpretation shall at all times attempt to predict future hazards in all areas of the underground excavation. Any potential hazards shall be recorded the Hazard Register., and presented and discussed with the relevant personnel. The Hazard Register and updates shall be authorized and signed by the Manager and updated regularly.

5.4.8 Document Control

All site SMPs shall have a document control process for:-

a. The site Strata Management Plan
b. Hazard Register
c. Strata Support Plan
d. Strata Support Rules

The control process shall require these documents to be reviewed and edited, if necessary, at least every six months.

5.4.9 Continuous Improvement

- **Review:** The controlled documents shall be reviewed and edited, if necessary, annually by a suitably competent person. Changes to the site controlled documents shall be ratified by the site Manager.
- **Audits:** Audits of the system and the Strata Support should be conducted regularly by an experienced person, preferably an experienced independent engineer.
  These Audits shall include but are not limited to SMP documentation, hazard mapping, review of miner's strata control understanding and feedback system, hardware audit (air pressure, bolters, resin storage, bolts, sets, lining etc.

5.5 Summary

About 50% of coal reserves in India are in seams thicker than 4.5m. Mass exploitation method by continuous miner for thick seam extraction with bord & pillar mining method and does not have goaf edge support. This method enhances coal recovery and practiced, as it cannot be afforded loss of national resources in terms of poor recovery of coal from thick seam. Mass exploitation by continuous miner method of working can be practiced in virgin thick seam as well as developed pillars in thick seam achieving higher percentage of recovery (70-80%) by using remote control continuous miner.
As the strata behavior of the roof rock of Indian coal measures poses problem due to its competent formations, large area of exposure of goaf are experienced during extraction causing serious strata disturbances. Geo-technical problems become worse while exploiting thick seam under Indian geo-mining conditions.

The main criterion of the success of depillaring a thick seam by continuous miner method is strata control. To analyze the existing system, real life data from different mines had been collected according to their strata control plans. It is observed that at the mines, records are being maintained to complied statutory obligations as described in permission conditions to work the panel. A thorough analysis with scientific approach will have control on strata while mining thick seam by continuous miner with bord & pillar mining method and it will yield higher percentage of recovery with highest safety. While analyzing the impact of depth on strata behaviour, it is observed that there are lots of things having significant relevance and importance such as area of exposure during first local fall, expected time of first major fall with regard to area of exposure and their dependence on depth of panel.

In this convergence study on strata behaviour in the field of thick seam mining by continuous miner method has a gainful understanding of the critical parameters involved in the success of the system. Significant dedication in collecting the information through various electronic roof parameter monitoring accessories have helped a lot in decision making for better control of the roof and to forecast on the following suggestions –

- Abnormal strata loading should be overcome by adopting suitable line of operation. A strict vigil on the performance of the adopted line of operation is the mandate.
- Goaf line velocity should be kept uniform as far as possible which will help to get an idea about the strata behaviour and progress of the line of extraction. This will further help to assess the strata condition vis-à-vis roof support at the point of breakage line.
- Besides depth, the geo-technical parameters including faults, folds and inherent weaknesses should be given due weightage prior to determining pillar and panel size and suitable line of operation.
- The numerical analysis as verified in this thesis work can well be referred in similar geo-mining conditions with varied depth of cover. This will help the management to take proper decision in time to tackle the immediate roof which constitutes the main part of the continuous miner method for mass exploitation of Coal.
- Coal transport and handling system should be matched with the mode of coal preparation in the panel. This will relieve the solid pillars of the static and dynamic strain.

Strata Management for continuous miner operations is illustrated in previous section in this chapter. The process involves obtaining field and laboratory data for rock mass characterization. The characterization assists in finding out an appropriate gallery width which in turns assists to find out a suitable width for the cutting drum for the continuous miner. Bieniawski's RMR and

CMRI-Geomechanical Classification are required to ascertain requirements of cut-out distance and gallery width. Economical operations for continuous miners dictate that the roadway width should be at least twice the drum width. The process then involves determination of roof deformation characteristics through rock mechanics instrumentation.

This valuable data not only decides the length of rock reinforcement measures but also used to simulated numerical models. Thereafter, simulated numerical models are used to predict overall rock mass response as a result of continuous mining operations. The above procedure is applied to NCPH colliery R-6 Mine, Chirimiri Area, SECL and the mine is successfully enjoying the benefits of the design model without a single instance of roof fall since inception of panel S-1 to end of coal recovery in this panel.

CHAPTER-6

CONCLUSIONS AND RECOMMENDATIONS

CONCLUSIONS AND RECOMMENDATIONS

6.0 General

This thesis aims to be a logical design methodology for coal mine extraction optimization under competent sandstone mine roof, illustrated by a comprehensive investigation and optimization study of the NCPH Colliery, R-6 mine, a shallow underground bord and pillar mine with a massive sandstone roof. This research is intended to guide mass exploitation of Coal in the mine and other coal mines globally in efforts to develop or optimize coal extraction and address the geo-mechanical challenges presented by massive sandstone roof. The key aspects and tools required to facilitate effective site characterization, ground support design, excavation stability, pillar design, environmental risk management and mining method optimization have been presented, as part of the design methodology.

The field work for this thesis was conducted at the NCPH Colliery, R-6 mine site, Chirimiri Area, M/s S.E.C.L. between July 2010 and December 2011. Rock mechanical properties test work and produce of extensometers, convergence recorder, Load cell & stress meter for this study was conducted at the Geo-mechanics data collected at the company laboratory and CMRI.
The objective of this thesis was to advance the design methods used to develop ground support systems, stable excavation spans and coal pillars and optimized coal extraction methods at shallow underground coal mines characterized by a massive sandstone roof.

Current design methodologies do not give adequate treatment of sandstone roof geology in the design of safe mining conditions for coal extraction, where significant engineering design input is required to address the complexities of this category of mine roof behavior. The importance of safety risk management is also not given adequate treatment in design methodologies at a time when neglect of this issue can severely undermine the credibility of a new project proposal in same area. This thesis critiqued available design methods. It defined the geo-mechanical and working considerations important to design and then applied analytical, empirical and numerical methods to support the final design of a ground support system, coal pillars and pillar extraction method for the R-6 mine. It is hoped that this research will be particularly valuable to the R-6 mine in its efforts to develop future coal mines with massive sandstone roof conditions.

Furthermore, this thesis is expected to benefit coal mine projects globally where similar geotechnical, geological and environmental risks are present, yet not adequately addressed with available tools and references. This chapter will review the important conclusions and recommendations as they relate to the study questions outlined in chapter 1. Contributions to the advancement of the state-of-the-art are also presented, followed by recommendations for future work.

6.1 Conclusions

Field studies have been conducted on strata behavior with respect to convergence during extraction of coal in a thick seam (4-6.5m) by continuous miner with diagonal slicing method at NCPH colliery, R-6 mine. In this method during depillaring operation no additional support installed (except break line support) as per our existing strata control technology. But operation of depillaring in thick seam (4-6.5m) is crucial as strata management because so many other factors influencing the mechanism of stability excavated area. Such as geo-mining parameters has variable due to extraction height and width changes in sequential manner. Our studies also concluded with reference to existing support system verification for ensuring safety.

Based on the results of the above mention field investigation and verification of numerical models with compare to empirical designed Roof bolt support system our study drawn following concluding remark:-

- During widening of galleries, a maximum of 16mm roof convergence is observed in NCPH Mine.
- Widening of galleries upto 6m has shown no considerable deformation or convergence but greater than 6m wide galleries has resulted in the formation of undulated roof and floor conditions in before modification of support system.
- Before this study, there was use of quick setting cement capsule in roof bolting. Based on Field observations, it has been modified to resin bolting so as to provide safer working conditions.
- Empirical estimates of support requirement for the NCPH mine roof suggest a conservative range of rock bolt lengths and pattern of support spacing, and a greater support load density and anchorage depth that have successfully been provided by the current installed ground support.

- The modified empirical support predictions prescribe 1.8 m for split and 2.4 m long roof bolts for original gallery and point-anchored rebar on 1.5 m center-to-center spacing, a design well suited to the roof conditions in NCPH mine of S.E.C.L.
- Resin roof bolts are finding more useful applications in underground coal mines, particularly un-tensioned, forged headed rebar, with lengths of 1.8m being the most accepted.
- Based on Numerical modeling results, the bolting was found efficient at a distance of 0.6m from the side of the pillar and 1.5m distance from the adjacent bolt.

6.2 Recommendations

For the safety of an underground pillar extraction process, it is always significant to conduct monitoring of strata control parameters. Systematic and suitable support system is one of the most important factors among different usually monitored strata control parameters during a depillaring operation. The investigations conducted under this study showed that the importance of monitoring of strata behavior during depillaring with diagonal slicing by continuous miner to address the safety issues. However, it is not always possible to conduct a field investigation in and around a depillaring face due to different techno-economic reasons. Further, it is also difficult to have any data of such monitoring for the first depillaring panel of a mine. For all situations, the results in this study may be used to improve support design and work schedule at the mines. Therefore, a quantitative assessment of value and strata behavior may provide a better idea about the convergence behavior of overlying roof strata, which can, ultimately, be used for different design of support purposes. On the basis of the scope and the limitations of this study, following recommendations are made for practical applications of the results obtained in this study:

- The proposed system can be applied also to gallery with larger cross sections (e.g. 40 to 50 m^2) and other shapes of the cross sections such as rectangular, trapezoidal etc.
- In weak rocks, rock bolting can be combined with other support measures like shotcrete, steel support or concrete. A similar numerical procedure might be applied to characterize the reinforcement effect in a quantitative manner.

- Our study correlated with empirical and numerical model for competence of support system in panel S-1 recommend 1.8m-long roof bolts on 0.75 m and 1.5 m center-to-center support spacing best suited for NCPH mine of S.E.C.L.
- A site specific systematic support system during depillaring, as per value of range of influence determined by the numerical model, may be more effective for strata control than a conventionally fixed range (30m or two pillars, whichever is more).

6.3 Limitations of the Study

In this study, an effort is made to cover strata behavior aspects with respect to convergence with existing support system during extraction of coal in a thick seam by continuous miner with diagonal slicing method at S-1panel NCPH Colliery R-6 Mine. However, the reported work in this thesis encounters some limitations, which are given below:

I. Research mentioned in this thesis is mainly limited to popular board and pillar developments and selected method of depillaring operation.

II. This study is limited to the mentioned a case study of continuous miner upto 6.5 m working height.

III. The range of depth of cover considered for the study is from 60 to 106m only.

IV. It is observed to be very difficult to collect every details of the site to strengthen the input parameters for the simulation work; therefore, established correlation between numerical and empirical formulations is used to supplement missing information and to calibrate the initial models.

V. The numerical model study is done using three dimensional finite difference code FLAC, mainly, due to memory and speed limitations of the available computing machine and availability of experience of using this package for Indian coal mines.

VI. To cover all the factors influencing during extraction of coal in a thick seam by continuous miner their entire ranges would have so many constraints. Therefore, simulation study for verification of existing support system is done only those selected parameters, which significantly influences in support performance of roof bolt. The study is limited for assessment of strata behavior within the domain of the convergence and load on support selected for this study.

VII. This study is done in depillaring panel with caving only

6.4 Future Scope of Research

Many parameters relevant for the study could not be covered due to practical problem of instrumentation and natural condition of the production panel. The following areas are considered to merit further study:

- Extension of present parameter investigation to cover other boundary conditions i.e. different depth of coal deposits and geo-mining conditions for better understanding of strata control specifics for mass exploitation of thick seam coal deposits. This will be more useful to development of simulation model with respect to strata behavior and Roof bolts support interaction.
- Availability of continuous (in time) monitoring data of the strata behavior and support interaction through a computer interfaced data logger may further value addition in modified support system.
- The effect of normalised parameters (like goaf treatment, height of extraction, gradient of seam etc.) should also be attempted to improve the system of support.

CHAPTER- 7

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APPENDICES

APPENDIX 1

ROOF CONVERGENCE MEASUREMENT DATA

1LE-S1-CONVERGENCE

DATE	GED	STATION- S1			STATION- S1A		
		READING	CON	CUM. CON	READING	CON	CUM. CON
2/7/2010	10	NS	NS	NS	NS	NS	NS
3/7/2010	10	26.4	0	0	18.0	0	0
4/7/2010	8	26.4	0	0	18.0	0	0
5/7/2010	8	26.4	0	0	18.0	0	0
6/7/2010	8	26.4	0	0	18.0	0	0
7/7/2010	6	26.4	0	0	19.9	1	1
8/7/2010	6	26.3	1	1	19.8	1	2
9/7/2010	6	26.2	1	2	19.8	0	2
10/7/2010	4	26.1	1	3	19.7	1	3
11/7/2010	4	26.0	1	4	19.6	1	4
12/7/2010	4	26.0	0	4	19.5	1	5
7/13/2010	4	25.9	1	5	19.5	0	5
7/14/2010	4	25.7	2	7	19.4	1	6
7/15/2010	4	25.7	0	7	19.4	0	6
7/16/2010	4	25.5	2	9	19.3	1	7
	GOAF						

1LE-S2-CONVERGENCE

DATE	GED	STATION- S1			STATION- S1A		
		READING	CON	CUM. CON	READING	CON	CUM. CON
2/7/2010	10	NS	NS	NS	NS	NS	NS

3/7/2010	10	26.4	0	0	18.0	0	0
4/7/2010	8	26.4	0	0	18.0	0	0
5/7/2010	8	26.4	0	0	18.0	0	0
6/7/2010	8	26.4	0	0	18.0	0	0
7/7/2010	6	26.4	0	0	19.9	1	1
8/7/2010	6	26.3	1	1	19.8	1	2
9/7/2010	6	26.2	1	2	19.8	0	2
10/7/2010	4	26.1	1	3	19.7	1	3
11/7/2010	4	26.0	1	4	19.6	1	4
12/7/2010	4	26.0	0	4	19.5	1	5
7/13/2010	4	25.9	1	5	19.5	0	5
7/14/2010	4	25.7	2	7	19.4	1	6
7/15/2010	4	25.7	0	7	19.4	0	6
7/16/2010	4	25.5	2	9	19.3	1	7
	GOAF						

1LE-S3-CON

DATE	GED	STATION- S3			STATION- S3A		
		READING	CON	CUM. CON	READING	CON	CUM. CON
8/7/2010	23	30.0	NS	NS	26.0	NS	NS
9/7/2010	23	30.0	0	0	26.0	0	0
10/7/2010	23	30.0	0	0	26.0	0	0
11/7/2010	23	30.0	0	0	26.0	0	0
12/7/2010	23	30.0	0	0	26.0	0	0
13/7/2010	23	29.9	1	1	25.9	1	1
14/7/2010	23	29.9	0	1	25.9	0	1
15/7/2010	21	29.9	0	1	25.9	0	1
16/7/2010	21	29.9	0	1	25.8	1	2
17/7/2010	21	29.9	0	1	25.8	0	2

18/7/2010	19	29.9	0	1	25.8	0	2
19/7/2010	19	29.8	0	1	25.8	0	2
20/7/2010	19	29.8	0	1	25.8	0	2
21/7/2010	19	29.8	0	1	25.7	1	3
22/7/2010	15	29.7	0	1	25.7	0	3
23/7/2010	15	29.6	1	2	25.6	1	4
24/7/2010	15	29.5	1	3	25.6	0	4
25/7/2010	15	29.5	0	3	25.5	1	5
26/7/2010	15	29.5	0	3	25.4	1	6
27/7/2010	15	29.5	0	3	25.4	0	6
28/7/2010	15	29.4	1	4	25.3	1	7
29/7/2010	15	29.4	0	4	25.3	0	7
30/7/2010	13	29.3	1	5	25.2	1	8
31/7/2010	13	29.1	2	7	25.1	1	9
1/8/2010	10	29.0	1	8	25.0	1	10
2/8/2010	10	28.9	1	9	24.9	1	11
3/8/2010	8	28.7	2	11	24.8	1	12
4/8/2010	8	28.6	1	12	24.7	1	13
5/8/2010	8	28.5	1	13	24.6	1	14
6/8/2010	8	28.5	0	13	24.6	0	14
7/8/2010	6	28.4	1	14	24.5	1	15
8/8/2010	4	28.3	1	15	24.4	1	16
9/8/2010	3	28.2	1	16	24.3	1	17
10/8/2010	3	28.2	0	16	24.3	0	17
11/8/2010	3	28.2	0	16	24.3	0	17
12/8/2010	3	28.1	1	17	24.2	1	18
		GOAF					

1LE-S4-CON

DATE	GED	STATION- S4			STATION- S4A		
		READING	CON	CUM. CON	READING	CON	CUM. CON
20/8/2010	18	40.3	NS	NS	34.0	NS	NS
21/8/2010	18	40.3	0	0	34.0	0	0
22/8/2010	16	40.2	0	0	34.0	0	0
23/8/2010	16	40.2	0	0	34.0	0	0
24/8/2010	16	40.2	0	0	34.0	0	0
25/8/2010	16	40.2	0	0	34.0	0	0
26/8/2010	14	40.2	0	0	34.0	0	0
27/8/2010	12	40.1	1	1	33.9	1	1
28/8/2010	11	39.9	1	2	33.8	1	2
29/8/2010	11	39.9	1	3	33.8	0	2
30/8/2010	11	39.9	0	3	33.8	0	2
31/8/2010	11	39.9	0	3	33.7	1	3
1/9/2010	11	39.9	0	3	33.7	0	3
2/9/2010	10	39.9	0	3	33.7	0	3
3/9/2010	10	39.8	1	4	33.6	1	4
4/9/2010	8	39.9	0	4	33.6	0	4
5/9/2010	8	39.9	0	4	33.5	1	5
6/9/2010	8	39.8	1	5	33.5	0	5
7/9/2010	6	39.8	0	5	33.4	1	6
8/9/2010	6	39.8	0	5	33.4	0	6
9/9/2010	4	39.6	2	7	33.4	1	7
10/9/2010	4	39.6	0	7	33.3	1	8
11/9/2010	4	39.5	1	8	33.3	0	8
12/9/2010	2	39.5	0	8	33.3	0	8
9/13/2010	2	39.4	1	9	33.2	1	9
9/14/2010	2	39.3	1	10	33.2	0	9
		GOAF					

1LE-S5-CON

DATE	GED	STATION- S5			STATION- S5A		
		READING	CON	CUM. CON	READING	CON	CUM. CON
9/7/2010	19	14.0	NS	NS	13.9	NS	NS
9/7/2010	19	14.0	0	0	13.9	0	0
9/8/2010	19	14.0	0	0	13.9	0	0
9/9/2010	17	14.0	0	0	13.9	0	0
9/10/2010	17	14.0	0	0	13.9	0	0
9/11/2010	17	14.0	0	0	13.9	0	0
9/12/2010	15	14.0	0	0	13.9	0	0
13/9/2010	15	14.0	0	0	13.9	0	0
14/9/2010	13	14.0	0	0	13.8	1	1
15/9/2010	13	14.0	0	0	13.8	0	1
16/9/2010	13	14.0	0	0	13.8	0	1
17/9/2010	11	14.0	0	0	13.8	0	1
18/9/2010	11	13.9	1	1	13.7	1	2
19/9/2010	11	13.9	0	1	13.7	0	2
20/9/2010	11	13.9	0	1	13.7	0	2
21/9/2010	8	13.8	1	2	13.6	1	3
9/22/2010	8	13.8	0	2	13.6	0	3
9/23/2010	8	13.8	0	2	13.6	0	3
9/24/2010	8	13.8	0	2	13.6	0	3
9/25/2010	8	13.7	1	3	13.5	1	4
9/26/2010	8	13.7	0	3	13.5	0	4
9/27/2010	8	13.7	0	3	13.4	1	5
9/28/2010	8	13.7	0	3	13.4	0	5
9/29/2010	8	13.7	0	3	13.4	0	5
9/30/2010	8	13.6	1	4	13.4	0	5
1/10/2010	8	13.6	0	4	13.3	1	6

2/10/2010	8	13.6	0	4	13.3	0	6
3/10/2010	8	13.6	0	4	13.3	0	6
4/10/2010	8	13.6	0	4	13.3	0	6
5/10/2010	8	13.6	0	4	13.3	0	6
6/10/2010	8	13.5	1	5	13.3	0	6
7/10/2010	8	13.5	0	5	13.3	0	6
8/10/2010	8	13.5	0	5	13.3	0	6
9/10/2010	8	13.5	0	5	13.3	0	6
10/10/2010	8	13.5	0	5	13.2	1	7
11/10/2010	8	13.4	1	6	13.1	1	8
12/10/2010	8	13.4	0	6	13.1	0	8
10/13/2010	8	13.4	0	6	13.1	0	8
10/14/2010	8	13.4	0	6	13.1	0	8
10/15/2010	8	13.3	1	7	13.1	0	8
10/16/2010	8	13.3	0	7	13.0	1	9
10/17/2010	8	13.3	0	7	13.0	0	9
10/18/2010	8	13.3	0	7	13.0	0	9
10/19/2010	8	13.3	0	7	13.0	0	9
20/10/2010	8	13.3	0	7	12.9	1	10
21/10/2010	8	13.2	1	8	12.8	1	11
22/10/2010	8	13.2	0	8	12.8	0	11
23/10/2010	8	13.2	0	8	12.8	0	11
24/10/2010	8	13.2	0	8	12.8	0	11
25/10/2010	8	13.1	1	9	12.8	0	11
26/10/2010	8	13.0	1	10	12.7	1	12
27/10/2010	8	12.9	1	11	12.7	0	12
28/10/2010	8	12.9	0	11	12.7	0	12
29/10/2010	8	12.8	1	12	12.7	0	12
30/10/2010	8	12.8	0	12	12.6	1	13
31/10/2010	8	12.7	1	13	12.5	1	14

11/1/2010	4	12.6	1	14	12.3	2	16
11/2/2010	4	12.5	1	15	12.3	0	16
11/3/2010	2	12.4	1	16	12.2	1	17
11/4/2010	2	12.4	0	16	12.1	1	18
	2	12.3	1	17	12.1	0	18
		GOAF					

1LE-S6-CON

DATE	GED	STATION- S6			STATION- S6A		
		READING	CON	CUM. CON	READING	CON	CUM. CON
27/9/2010	24	13.0	NS	NS	15.3	NS	NS
28/9/2010	22	13.0	0	0	15.2	1	1
29/9/2010	22	13.0	0	0	15.2	0	1
30/9/2010	22	13.0	0	0	15.2	0	1
1/10/2010	22	13.0	0	0	15.2	0	1
2/10/2010	22	12.9	1	1	15.2	0	1
3/10/2010	22	12.9	0	1	15.2	0	1
4/10/2010	22	12.9	0	1	15.2	0	1
5/10/2010	22	12.9	0	1	15.2	0	1
6/10/2010	22	12.9	0	1	15.2	0	1
7/10/2010	22	12.8	1	2	15.2	0	1
8/10/2010	22	12.8	0	2	15.2	0	1
9/10/2010	22	12.7	1	3	15.1	1	2
10/10/2010	22	12.7	0	3	15.1	0	2
11/10/2010	22	12.7	0	3	15.1	0	2
12/10/2010	18	12.7	0	3	15.1	0	2
13/10/2010	18	12.7	0	3	15.1	0	2
14/10/2010	18	12.7	0	3	15.1	0	2

15/10/2010	18	12.6	1	4	15.1	0	2
16/10/2010	18	12.6	0	4	15.1	0	2
17/10/2010	18	12.6	0	4	15.0	1	3
18/10/2010	18	12.5	1	5	15.0	0	3
19/10/2010	18	12.5	0	5	15.0	0	3
20/10/2010	18	12.5	0	5	14.9	1	4
21/10/2010	18	12.5	0	5	14.9	0	4
22/10/2010	18	12.5	0	5	14.9	0	4
23/10/2010	18	12.4	1	6	14.9	0	4
24/10/2010	18	12.4	0	6	14.8	1	5
25/10/2010	18	12.4	0	6	14.8	0	5
26/10/2010	18	12.4	0	6	14.8	0	5
27/10/2010	18	12.4	0	6	14.8	0	5
28/10/2010	18	12.4	0	6	14.7	1	6
29/10/2010	18	12.3	1	7	14.7	0	6
30/10/2010	18	12.3	0	7	14.7	0	6
31/10/2010	18	12.3	0	7	14.7	0	6
1/11/2010	18	12.3	0	7	14.7	0	6
2/11/2010	18	12.2	1	8	14.7	0	6
3/11/2010	18	12.1	1	9	14.7	0	6
4/11/2010	18	12.0	1	10	14.6	1	7
5/11/2010	18	12.0	0	10	14.6	0	7
6/11/2010	18	12.0	0	10	14.6	0	7
7/11/2010	18	12.0	0	10	14.6	0	7
8/11/2010	17	11.9	1	11	14.6	0	7
9/11/2010	17	11.7	2	13	14.5	1	8
11/10/2010	16	11.6	1	14	14.3	2	10
11/11/2010	16	11.6	0	14	14.2	1	11
11/12/2010	16	11.5	1	15	14.2	0	11
13/11/2010	14	11.5	0	15	14.2	0	11

14/11/2010	14	11.4	1	16	14.2	0	11
15/11/2010	14	11.3	1	17	14.1	1	12
16/11/2010	2	11.3	0	17	14.1	0	12
17/11/2010	2	11.2	1	18	14.0	1	13
		GOAF					

2LE-S1-CON

DATE	GED	STATION- S1			STATION- S1A		
		READING	CON	CUM. CON	READING	CON	CUM. CON
7/4/2010	10	23.7	NS	NS	25.8	NS	NS
7/5/2010	8	23.6	1	1	25.7	1	1
7/6/2010	8	23.5	1	2	25.6	1	2
7/7/2010	8	23.4	1	3	25.6	1	3
7/8/2010	6	23.3	1	4	25.5	1	4
7/9/2010	6	23.2	1	5	25.5	0	4
7/10/2010	6	23.1	1	6	25.4	1	5
7/11/2010	6	23.0	1	7	25.4	0	5
7/12/2010	6	22.9	1	8	25.2	2	7
14/7/2010	6	22.8	1	9	25.1	1	8
15/7/2010	6	22.8	0	9	25.0	0	8
16/7/2010	4	22.9	1	10	24.9	1	9
17/7/2010	4	22.8	1	11	24.8	1	10
18/7/2010	4	22.7	1	12	24.7	1	11
19/7/2010	4	22.7	0	12	24.6	1	12
20/7/2010	4	22.7	0	12	24.5	1	13
21/7/2010	2	22. 5	2	14	24.4	1	14
	GOAF						

2LE-S2-CON

DATE	GED	STATION-S2			STATION- S2A		
		READING	CON	CUM. CON	READING	CON	CUM. CON
21/9/2010	20	32.6	NS	NS	27.8	NS	NS
22/9/2010	18	32.6	0	1	27.8	0	0
23/9/2010	18	32.6	0	1	27.7	1	1
24/9/2010	18	32.6	0	1	27.7	0	1
26/9/2010	16	32.5	1	2	27.6	1	2
27/9/2010	16	32.5	0	2	27.6	0	2
28/9/2010	16	32.4	1	3	27.6	0	2
29/9/2010	16	32.4	0	3	27.6	0	2
30/9/2010	16	32.3	1	4	27.5	1	3
1/10/2010	16	32.2	1	5	27.5	0	3
2/10/2010	16	32.2	0	5	27.4	1	4
3/10/2010	14	32.1	1	6	27.3	1	5
4/10/2010	14	32.0	1	7	27.2	1	6
5/10/2010	14	32.0	0	7	27.1	1	7
6/10/2010	14	31.9	1	8	27.1	0	7
7/10/2010	14	31.9	0	8	27.1	0	7
8/10/2010	12	31.8	1	9	27.0	1	8
9/10/2010	12	31.7	1	10	26.9	1	9
10/10/2010	12	31.6	1	11	26.8	1	10
11/10/2010	10	31.5	1	12	26.7	1	11
12/10/2010	10	31.5	0	12	26.7	0	11
10/13/2010	8	31.3	2	14	26.6	1	12
10/14/2010	8	31.3	0	14	26.5	1	13
10/15/2010	8	31.3	0	14	26.4	1	14
10/16/2010	6	31.2	1	15	26.2	2	16
10/17/2010	6	31.1	1	16	26.2	0	16

10/18/2010	3	31.0	1	17	26.2	0	16
10/19/2010	3	30.9	1	18	26.2	0	16
10/20/2010	3	30.8	1	19	26	2	18
	GOAF						

2LE-S3-CON

DATE	GED	STATION- S3			STATION-S3A		
		READING	CON	CUM. CON	READING	CON	CUM. CON
12/11/2010	22	26.5	NS	NS	25.9	NS	NS
12/12/2010	22	26.5	0	0	25.9	0	0
13/12/2010	20	18.0	PD	PD	25.8	1	1
14/12/2010	20	17.9	1	0	25.8	0	1
15/12/2010	18	17.9	0	0	25.8	0	1
16/12/2010	18	17.9	0	0	25.8	0	1
17/12/2010	18	17.8	1	1	25.8	0	1
18/12/2010	16	17.8	0	1	25.7	1	2
19/12/2010	16	17.8	0	1	25.7	0	2
20/12/2010	13	17.7	1	2	25.7	0	2
21/12/2010	13	17.7	0	2	25.6	1	3
22/12/2010	13	17.6	1	3	25.5	1	4
23/12/2010	10	16.7	1	4	25.4	1	5
24/12/2010	10	16.5	0	4	25.4	0	5
25/12/2010	10	16.4	1	5	25.4	0	5
26/12/2010	10	16.4	0	5	25.4	0	5
27/12/2010	10	16.4	0	5	25.3	1	6
28/12/2010	10	16.3	1	6	25.2	1	7
12/29/2010	8	16.3	1	7	25.1	1	8
30/12/2010	7	16.2	1	8	25.1	0	8

31/12/2010	7	16.1	1	9	25.1	0	8
1/1/2011	7	16.0	1	10	25.1	0	8
2/1/2011	7	15.8	2	12	25.0	1	9
3/1/2011	6	15.6	2	14	24.9	1	10
4/1/2011	6	15.6	0	14	24.8	1	11
5/1/2011	3	15.5	1	15	24.7	1	12
6/1/2011	3	15.5	0	15	24.6	1	13
7/1/2011	3	15.4	1	16	23.7	1	14
8/1/2011	3	15.3	1	17	23.6	1	15
9/1/2011	3	15.3	0	17	23.6	0	15
10/1/2011	3	15.3	0	17	23.5	0	15
11/1/2011	3	15.3	0	17	23.4	1	16
12/1/2011	3	15.2	1	18	23.4	0	16
13/1/2011	3	15.2	0	18	23.4	0	16

2LE-S4-CON

DATE	GED	STATION- S4			STATION- S4A		
		READING	CON	CUM. CON	READING	CON	CUM. CON
21/10/2010	21	16.3	NS	NS	10.1	NS	NS
22/10/2010	21	16.3	0	0	10.1	0	0
23/10/2010	21	16.2	0	0	10.0	0	0
24/10/2010	21	16.2	0	0	10.0	0	0
25/10/2010	21	16.2	0	0	10.0	0	0
26/10/2010	21	16.1	1	1	10.0	0	0
27/10/2010	19	16.0	1	2	9.8	1	1
28/10/2010	18	15.9	1	3	9.7	1	2
29/10/2010	18	15.9	0	3	9.7	0	2
30/10/2010	18	15.8	1	4	9.6	1	3
31/10/2010	17	15.7	1	5	9.6	0	3

1/11/2010	17	15.6	1	6	9.5	1	4
2/11/2010	17	15.5	1	7	9.5	0	4
3/11/2010	14	15.5	0	7	9.5	0	4
4/11/2010	14	15.4	1	8	9.3	2	6
5/11/2010	14	15.1	1	9	9.2	1	7
6/11/2010	14	15.1	0	9	9.1	1	8
7/11/2010	14	15.0	1	10	9.1	0	8
8/11/2010	14	14.9	1	11	9.0	1	9
9/11/2010	14	14.9	0	11	9.0	0	9
10/11/2010	14	14.9	0	11	8.9	1	10
11/11/2010	14	14.9	0	11	8.8	1	11
12/11/2010	14	14.9	0	11	8.8	0	11
13/11/2010	14	14.9	0	11	8.6	1	12
14/11/2010	14	14.8	1	12	8.5	1	13
15/11/2010	13	14.7	1	13	8.3	2	15
16/11/2010	13	14.6	1	14	8.1	2	17
18/11/2010	11	14.6	0	14	8.0	1	18
19/11/2010	11	14.5	1	15	7.9	1	19
20/11/2010	8	14.4	1	16	7.8	1	20
21/11/2010	8	14.3	1	17	7.7	1	21
22/11/2010	8	14.2	1	18	7.7	0	21
23/11/2010	8	14.1	1	19	7.6	1	22
24/11/2010	5	13.9	2	21	7.4	2	24
25/11/2010	5	13.7	2	23	7.3	1	25
26/11/2010	5	13.5	2	25	7.1	2	27
27/11/2010	3	13.3	2	27	6.9	2	29
28/11/2010		GOAF					

2LE-S4-CON

DATE	GED	STATION- S5			STATION- S5A		
		READING	CON	CUM. CON	READING	CON	CUM. CON
14/11/2010	22	23.2	NS	NS	19.6	NS	NS
16/11/2010	22	23.0	0	2	19.4	0	0
17/11/2010	20	23.0	0	2	19.3	1	1
18/11/2010	20	22.9	1	3	19.2	1	2
19/11/2010	20	22.9	0	3	19.2	0	2
20/11/2010	17	22.8	1	4	19.0	2	4
21/11/2010	17	22.8	0	4	18.9	1	5
22/11/2010	17	22.7	1	5	18.8	1	6
23/11/2010	17	22.6	1	6	18.7	1	7
24/11/2010	15	22.5	1	7	18.6	1	8
25/11/2010	15	22.4	1	8	18.6	0	8
26/11/2010	15	22.3	1	9	18.4	2	10
27/11/2010	13	22.2	1	10	18.3	1	11
28/11/2010	12	22.1	1	11	18.1	2	13
29/11/2010	11	21.9	2	13	18.1	0	13
30/11/2010	10	21.7	2	15	17.9	2	15
12/1/2010	10	21.6	1	16	17.7	2	17
12/2/2010	10	21.5	1	17	17.5	2	19
12/3/2010	8	21.4	1	18	17.3	2	21
12/4/2010	8	21.3	1	19	17.3	0	21
12/5/2010	6	21.3	0	19	17.1	2	23
12/6/2010	6	21.1	2	21	17.0	1	24
12/7/2010	6	20.9	2	23	17.0	0	24
12/8/2010	4	20.8	1	24	16.9	1	25
12/9/2010	4	20.7	1	25	16.9	0	25
12/10/2010	4	20.6	1	26	16.8	1	26

12/11/2010	4	20.5	1	27	16.7	1	27
12/12/2010	4	20.4	1	28	16.6	1	28
13/12/2010	4	20.3	1	29	16.5	1	29
14/12/2010	4	20.2	1	30	16.4	1	30
15/12/2010	4	20.1	1	31	16.3	1	31
16/12/2010	4	20.1	0	31	16.3	0	31
17/12/2010	4	20.0	1	32	16.3	0	31
18/12/2010	4	20.0	0	32	16.3	0	31
19/12/2010	4	20.0	0	32	16.3	0	31
20/12/2010	4	20.0	0	32	16.1	2	33
21/12/2010	4	20.0	0	32	16.1	0	33
22/12/2010	4	19.9	1	33	16.0	1	34
23/12/2010	4	19.9	0	33	16.0	0	34
24/12/2010	4	19.9	0	33	16.0	0	34
25/12/2010	4	19.8	1	34	15.9	1	35
26/12/2010	4	19.8	0	34	15.9	0	35
		GOAF					

2LE-S6-CON

DATE	GED	STATION- S6			STATION- S6A		
		READING	CON	CUM. CON	READING	CON	CUM. CON
28/11/2010	26	31.5	NS	NS	32.8	NS	NS
29/11/2010	26	31.5	0	0	32.8	0	0
30/11/2010	25	31.4	1	1	32.7	1	1
12/1/2010	25	31.4	0	1	32.6	1	2
12/2/2010	25	31.3	1	2	32.6	0	2
12/3/2010	23	31.3	0	2	32.5	1	3
12/4/2010	23	31.2	1	3	32.4	1	4
12/5/2010	21	31.2	0	3	32.4	0	4
12/6/2010	21	31.1	1	4	32.2	2	6
12/7/2010	21	31.1	0	4	32.0	2	8
12/8/2010	19	30.9	2	6	32.0	0	8
12/9/2010	19	30.8	1	7	31.8	2	10
12/10/2010	19	30.7	1	8	31.8	0	10
12/11/2010	19	30.6	1	9	31.7	1	11
12/12/2010	19	30.5	1	10	31.6	1	12
13/12/2010	19	30.4	1	11	31.6	0	12
14/12/2010	19	30.3	1	12	31.6	0	12
15/12/2010	19	30.3	0	12	31.6	0	12
16/12/2010	19	30.3	0	12	31.6	0	12
17/12/2010	19	30.3	0	12	31.5	1	13
18/12/2010	19	30.2	1	13	31.5	0	13
19/12/2010	19	30.2	0	13	31.5	0	13
20/12/2010	19	30.1	1	14	31.5	0	13
21/12/2010	19	30.1	0	14	31.5	0	13
22/12/2010	19	30.1	0	14	31.5	0	13
23/12/2010	19	30.0	1	15	31.4	1	14

24/12/2010	19	30.0	0	15	31.4	0	14
25/12/2010	19	29.9	1	16	31.3	1	15
26/12/2010	16	29.9	0	16	31.3	0	15
27/12/2010	16	29.9	0	16	31.3	0	15
28/12/2010	16	29.9	0	16	31.3	0	15
29/12/2010	16	29.8	1	17	31.3	0	15
30/12/2010	16	29.8	0	17	31.3	0	15
31/12/2010	16	29.7	1	18	31.2	1	16
1/1/2011	16	29.7	0	18	31.1	1	17
2/1/20111	16	29.6	1	19	31.1	0	17
3/1/2011	16	29.6	0	19	31.1	0	17
4/1/2011	16	29.6	0	19	31.1	0	17
5/1/2011	16	29.6	0	19	31.1	0	17
6/1/2011	16	29.6	0	19	31.0	1	18
7/1/2011	16	29.5	1	20	31.0	0	18
8/1/2011	16	29.4	1	21	31.0	0	18
9/1/2011	16	29.4	0	21	31.0	0	18
10/1/2011	16	29.4	0	21	31.0	0	18
11/1/2011	16	29.3	1	22	31.0	0	18
12/1/2011	16	29.3	0	22	30.9	1	19
13/1/2011	16	29.2	1	23	30.9	0	19
14/1/2011	16	29.2	0	23	30.9	0	19
15/1/2011	16	29.2	0	23	30.9	0	19
16/1/2011	16	29.2	0	23	30.9	0	19
17/1/2011	16	29.1	1	24	30.8	1	20
18/1/2011	16	29.1	0	24	30.8	0	20
19/1/2011	16	29.1	0	24	30.8	0	20
20/1/2011	16	29.0	1	25	30.7	1	21
21/1/2011	16	28.9	1	26	30.6	1	22
22/1/2011	16	28.7	2	28	30.4	2	24

23/1/2011	7	28.6	1	29	30.3	1	25
24/1/2011	7	28.6	0	29	30.3	0	25
25/1/2011	7	28.6	0	29	30.2	1	26
26/1/2011	7	28.5	1	30	30.2	0	26
27/1/2011	7	28.5	0	30	30.1	1	27
28/1/2011	3	28.4	1	31	30.1	0	27
29/1/2011	3	28.4	0	31	30.1	0	27
30/1/2011	3	28.4	0	31	30.1	0	27
31/1/2011	3	28.3	1	32	30.0	1	28
		GOAF					

2LE-S7-CON

DATE	GED	STATION- S7			STATION- S7A		
		READING	CON	CUM. CON	READING	CON	CUM. CON
23/1/2011	17	39.3	NS	NS	35.5	NS	NS
24/1/2011	17	39.2	1	0	35.5	0	0
25/1/2011	17	39.2	0	0	35.5	0	0
26/1/2011	17	39.2	0	0	35.5	0	0
27/1/2011	17	39.1	1	1	35.4	1	1
28/1/2011	17	39.0	1	2	35.3	1	2
29/1/2011	13	38.8	2	4	35.2	1	3
30/1/2011	13	38.8	0	4	35.2	0	3
31/1/2011	13	38.7	1	5	35.1	1	4
2/1/2011	13	38.5	2	7	35.0	1	5
2/2/2011	13	38.4	1	8	34.9	1	6
2/3/2011	9	38.3	1	9	34.8	1	7

2/4/2011	6	38.2	1	10	34.7	1	8
2/5/2011	6	38.1	1	11	34.6	1	9
2/6/2011	5	37.9	2	13	34.4	2	11
2/7/2011	5	37.9	0	13	34.4	0	11
2/8/2011	5	37.8	1	14	34.3	1	12
2/9/2011	5	37.7	1	15	34.2	1	13
		GOAF					

2LE-S8-CON

DATE	GED	STATION- S8			STATION- S8A		
		READING	CON	CUM. CON	READING	CON	CUM. CON
1/2/2011	21	6.6	NS	NS	3.8	NS	NS
2/2/2011	21	6.6	1	0	3.8	0	0
3/2/2011	18	6.5	1	1	3.7	1	1
4/2/2011	18	6.4	1	2	3.6	1	2
5/2/2011	17	6.4	0	2	3.5	1	3
6/2/2011	17	6.4	0	2	3.5	0	3
7/2/2011	17	6.3	1	3	3.2	3	6
8/2/2011	17	6.3	0	3	3.1	1	7
9/2/2011	17	6.3	0	3	2.9	2	9
10/2/2011	14	6.2	1	4	2.9	0	9
11/2/2011	14	6.0	2	6	2.9	0	9
12/2/2011	11	6.0	0	6	2.9	0	9
13/2/2011	11	5.8	2	8	2.7	2	11
14/2/2011	11	5.6	2	10	2.7	0	11

15/2/2011	11	5.6	0	10	2.5	2	13
16/2/2011	8	5.4	2	12	2.3	2	15
17/2/2011	8	5.3	1	13	2.2	1	16
18/2/2011	8	5.3	0	13	2.2	0	16
19/2/2011	6	5.2	1	14	2.1	1	17
20/2/2011	6	5.1	1	15	2.0	1	18
21/2/2011	6	5.1	0	15	1.8	2	20
22/2/2011	4	5.0	1	16	1.8	0	20
23/2/2011	4	4.8	2	18	1.7	1	21
		GOAF					

3LE-S1-CON

DATE	GED	STATION- S1			STATION- S1A		
7/11/2010		READING	CON	CUM. CON	READING	CON	CUM. CON
7/11/2010	10	23.7	NS	NS	25.8	NS	NS
7/12/2010	8	23.6	1	1	25.8	0	0
13/7/2010	8	23.5	1	2	25.7	1	1
14/7/2010	8	23.4	1	3	25.5	2	3
15/7/2010	6	23.3	1	4	25.4	1	4
16/7/2010	6	23.2	1	5	25.3	1	5
17/7/2010	6	23.1	1	6	25.2	1	6
18/7/2010	6	23.0	1	7	25.1	1	7
19/7/2010	6	22.9	1	8	25.0	1	8
20/7/2010	6	22.8	1	9	24.9	1	9
21/7/2010	6	22.7	1	10	24.8	1	10
22/7/2010	4	22.6	1	11	24.7	1	11
23/7/2010	4	22.5	1	12	24.6	1	12
24/7/2010	4	22.4	1	13	24.5	1	13
25/7/2010	4	22.2	2	15	24.4	1	14
26/7/2010	4	22.1	1	16	24.3	1	15

22/7/2010	2	22.0	1	17	24.2	1	16
	GOAF						

3LE-S2-CON

DATE	GED	STATION-S2			STATION- S2A		
		READING	CON	CUM. CON	READING	CON	CUM. CON
7/11/2010	20	32.6	**NS**	**NS**	27.8	**NS**	**NS**
7/12/2010	18	32.6	0	0	27.8	0	0
13/7/2010	18	32.5	1	1	27.7	1	1
14/7/2010	18	32.5	0	1	27.7	0	1
15/7/2010	16	32.4	1	2	27.6	1	2
16/7/2010	16	32.2	2	4	27.6	0	2
17/7/2010	16	32.1	1	5	27.6	0	2
18/7/2010	16	32.1	0	5	27.6	0	2
19/7/2010	16	32.0	1	6	27.5	1	3
20/7/2010	16	31.9	1	7	27.5	0	3
21/7/2010	16	31.9	0	7	27.2	3	6
22/7/2010	14	31.8	1	8	27.1	1	7
23/7/2010	14	31.6	2	10	27.0	1	8
24/7/2010	14	31.6	0	10	26.9	1	9
25/7/2010	14	31.5	1	11	26.9	0	9
26/7/2010	14	31.5	0	11	26.9	0	9
27/7/2010	12	31.4	1	12	26.8	1	10
28/7/2010	12	31.1	3	15	26.7	1	11
29/7/2010	12	31.0	1	16	26.6	1	12
30/7/2010	10	30.8	2	18	26.5	1	13
31/7/2010	10	30.8	0	18	26.5	0	13
8/1/2010	8	30.5	3	21	26.2	3	16
8/2/2010	8	30.5	0	21	26.1	1	17
8/3/2010	8	30.5	0	21	26	1	18

8/4/2010	6	30.3	2	23	25.7	3	21
8/5/2010	6	30.1	2	25	25.7	0	21
8/6/2010	3	29.9	2	27	25.7	0	21
8/7/2010	3	29.8	1	28	25.7	0	21
8/8/2010	3	29.6	2	30	25.5	2	23
	GOAF						

3LE-S3-CON

DATE	GED	STATION- S3			STATION-S3A		
		READING	CON	CUM. CON	READING	CON	CUM. CON
11/7/2010	22	26.5	NS	NS	25.9	NS	NS
12/7/2010	22	26.5	0	0	25.9	0	0
13/7/2010	20	18.0	PD	PD	25.9	1	1
14/7/2010	20	17.9	1	1	25.9	0	1
15/7/2010	18	17.9	0	1	25.8	1	2
16/7/2010	18	17.9	0	1	25.7	0	2
17/7/2010	18	17.8	1	2	25.7	0	2
18/7/2010	16	17.7	1	3	25.6	1	3
19/7/2010	16	17.6	1	4	25.5	1	4
20/7/2010	13	17.0	1	5	25.5	0	4
21/7/2010	13	17.5	0	5	25.4	1	5
22/7/2010	13	17.3	2	7	25.3	1	6
23/7/2010	10	17.2	1	8	25.2	1	7
24/7/2010	10	16.5	1	9	25.2	0	7
25/7/2010	10	16.3	1	10	25.1	1	8
26/7/2010	10	16.3	0	10	25.0	1	9
27/7/2010	10	16.1	2	12	24.9	1	10
28/7/2010	10	15.9	2	14	24.8	1	11
29/7/2010	8	15.8	1	15	24.7	1	12

30/7/2010	7	15.5	2	17	24.6	1	13
31/7/2010	7	15.2	1	18	24.5	1	14
8/1/2010	7	15.1	1	19	24.4	1	15
8/2/2010	7	15.0	1	20	24.3	1	16
8/3/2010	6	14.8	2	22	24.2	1	17
8/4/2010	6	14.7	1	23	24.1	1	18
8/5/2010	3	14.6	1	24	24.0	1	19
8/6/2010	3	14.4	2	26	23.8	2	21
8/7/2010	3	14.1	1	27	23.7	1	22
8/8/2010	3	14.0	1	28	23.6	1	23
8/9/2010	3	13.8	2	30	23.6	0	23
8/10/2010	3	13.8	0	30	23.5	1	24
8/11/2010	3	13.7	1	31	23.4	1	25
8/12/2010	3	13.6	1	32	23.4	0	25
13/8/2010	3	13.6	0	32	23.4	0	25
		GOAF					

3LE-S4-CON

DATE	GED	STATION- S4			STATION- S4A		
		READING	CON	CUM. CON	READING	CON	CUM. CON
7/11/2010	21	16.3	NS	NS	10.1	NS	NS
7/12/2010	21	16.3	0	0	10.1	0	0
13/7/2010	21	16.3	0	0	10.0	1	1
14/7/2010	21	16.3	0	0	10.0	0	1
15/7/2010	21	16.3	0	0	10.0	0	1
16/7/2010	21	16.2	1	1	10.0	0	1
17/7/2010	19	15.9	1	2	9.8	2	3

18/7/2010	18	15.8	1	3	9.7	1	4
19/7/2010	18	15.8	0	3	9.7	0	4
20/7/2010	18	15.7	1	4	9.6	1	5
21/7/2010	17	15.7	0	4	9.6	0	5
22/7/2010	17	15.6	1	5	9.5	1	6
23/7/2010	17	15.5	1	6	9.5	0	6
24/7/2010	14	15.5	0	6	9.5	0	6
25/7/2010	14	15.4	1	7	9.3	2	8
26/7/2010	14	15.1	3	10	9.2	1	9
27/7/2010	14	15.1	0	10	9.1	1	10
28/7/2010	14	15.0	1	11	9.1	0	10
29/7/2010	14	14.8	2	13	9.0	1	11
30/7/2010	14	14.8	0	13	9.0	0	11
31/7/2010	14	14.8	0	13	8.9	1	12
8/1/2010	14	14.7	1	14	8.8	1	13
8/2/2010	14	14.6	1	15	8.8	0	13
8/3/2010	14	14.5	1	16	8.6	2	15
8/4/2010	14	14.4	1	17	8.5	1	16
8/5/2010	13	14.1	3	20	8.2	3	19
8/6/2010	13	13.8	3	23	8.0	2	21
8/7/2010	11	13.7	1	24	7.9	1	22
8/8/2010	11	13.6	1	25	7.8	1	23
8/9/2010	8	13.4	2	27	7.6	2	25
8/10/2010	8	13.3	1	28	7.5	1	26
8/11/2010	8	13.2	1	29	7.5	0	26
8/12/2010	8	12.9	3	32	7.2	3	29
13/8/2010	5	12.8	1	33	6.9	1	30
14/8/2010	5	12.7	1	34	6.8	1	31
15/8/2010	5	12.2	1	35	6.6	2	33
16/8/2010	3	11.9	2	37	6.5	1	34

17/8/2010		GOAF					

3LE-S5-CON

DATE	GED	STATION- S5			STATION- S5A		
		READING	CON	CUM. CON	READING	CON	CUM. CON
11/8/2010	22	23.1	NS	NS	19.6	NS	NS
12/8/2010	22	23.0	1	1	19.4	2	2
13/8/2010	20	22.9	1	2	19.2	2	4
14/8/2010	20	22.8	1	3	19.1	1	5
15/8/2010	20	22.8	0	3	19.1	0	5
16/8/2010	17	22.7	1	4	18.9	2	7
17/8/2010	17	22.6	1	5	18.8	1	8
18/8/2010	17	22.5	1	6	18.6	2	10
19/8/2010	17	22.3	2	8	18.4	2	12
20/8/2010	15	22.2	1	9	18.3	1	13
21/8/2010	15	22.1	1	10	18.3	0	13
22/8/2010	15	22.0	1	11	18.0	3	16
23/8/2010	13	21.8	2	13	17.9	1	17
24/8/2010	12	21.7	1	14	17.7	2	19
25/8/2010	11	21.6	1	15	17.7	0	19
26/8/2010	10	21.4	2	17	17.5	2	21
27/8/2010	10	21.3	1	18	17.4	1	22
28/8/2010	10	21.2	1	19	17.3	1	23
29/8/2010	8	21.1	1	20	17.2	1	24
30/8/2010	8	21.0	1	21	17.2	0	24
31/8/2010	6	20.9	1	22	17.0	2	26
9/1/2010	6	20.7	2	24	16.9	1	27
9/2/2010	6	20.5	2	26	16.9	0	27
9/3/2010	4	20.4	1	27	16.8	1	28

9/4/2010	4	20.3	1	28	16.8	0	28
9/5/2010	4	20.2	1	29	16.7	1	29
9/6/2010	4	20.1	1	30	16.6	1	30
9/7/2010	4	20.0	1	31	16.5	1	31
8/92010	4	19.8	2	33	16.4	1	32
9/9/2010	4	19.7	1	34	16.3	1	33
9/10/2010	4	19.6	1	35	16.2	1	34
9/11/2010	4	19.6	0	35	16.2	0	34
9/12/2010	4	19.5	1	36	16.2	0	34
13/9/2010	4	19.5	0	36	16.2	0	34
14/9/2010	4	19.5	0	36	16.2	0	34
15/9/2010	4	19.5	0	36	16.1	1	35
16/9/2010	4	19.5	0	36	16.1	0	35
17/9/2010	4	19.4	1	37	16.0	1	36
18/9/2010	4	19.4	0	37	16.0	0	36
19/9/2010	4	19.3	1	38	16.0	0	36
20/9/2010	4	19.2	1	39	15.9	1	37
21/9/2010	4	19.2	0	39	15.9	0	37
		GOAF					

3LE-S6-CON

DATE	GED	STATION- S6			STATION- S6A		
		READING	CON	CUM. CON	READING	CON	CUM. CON
14/8/2010	26	31.5	NS	NS	32.8	NS	NS
15/8/2010	26	31.5	0	0	32.8	0	0
16/8/2010	25	31.4	1	1	32.7	1	1
17/8/2010	25	31.4	0	1	32.6	1	2
18/8/2010	25	31.3	1	2	32.6	0	2
19/8/2010	23	31.3	0	2	32.5	1	3

20/8/2010	23	31.2	1	3	32.4	1	4
21/8/2010	21	31.2	0	3	32.4	0	4
22/8/2010	21	31.1	1	4	32.2	2	6
23/8/2010	21	31.1	0	4	32.0	2	8
24/8/2010	19	30.9	2	6	32.0	0	8
25/8/2010	19	30.8	1	7	31.8	2	10
26/8/2010	19	30.7	1	8	31.8	0	10
27/8/2010	19	30.6	1	9	31.7	1	11
28/8/2010	19	30.5	1	10	31.6	1	12
29/8/2010	19	30.4	1	11	31.6	0	12
30/8/2010	19	30.3	1	12	31.6	0	12
31/8/2010	19	30.3	0	12	31.6	0	12
1/9/2010	19	30.3	0	12	31.6	0	12
2/9/2010	19	30.3	0	12	31.5	1	13
3/9/2010	19	30.2	1	13	31.5	0	13
4/9/2010	19	30.2	0	13	31.5	0	13
5/9/2010	19	30.1	1	14	31.5	0	13
6/9/2010	19	30.1	0	14	31.5	0	13
7/9/2010	19	30.1	0	14	31.5	0	13
8/9/2010	19	30.0	1	15	31.4	1	14
9/9/2010	19	30.0	0	15	31.4	0	14
10/9/2010	19	29.9	1	16	31.3	1	15
11/9/2010	16	29.9	0	16	31.3	0	15
12/9/2010	16	29.9	0	16	31.3	0	15
13/9/2010	16	29.9	0	16	31.3	0	15
14/9/2010	16	29.8	1	17	31.3	0	15
15/9/2010	16	29.8	0	17	31.3	0	15
16/9/2010	16	29.7	1	18	31.2	1	16
17/9/2010	16	29.7	0	18	31.1	1	17
18/9/2010	16	29.6	1	19	31.1	0	17

19/9/2010	16	29.6	0	19	31.1	0	17
20/9/2010	16	29.6	0	19	31.1	0	17
21/9/2010	16	29.6	0	19	31.1	0	17
22/9/2010	16	29.6	0	19	31.0	1	18
23/9/2010	16	29.5	1	20	31.0	0	18
24/9/2010	16	29.4	1	21	31.0	0	18
25/9/2010	16	29.4	0	21	31.0	0	18
26/9/2010	16	29.4	0	21	31.0	0	18
27/9/2010	16	29.3	1	22	31.0	0	18
28/9/2010	16	29.3	0	22	30.9	1	19
29/9/2010	16	29.2	1	23	30.9	0	19
30/9/2010	16	29.2	0	23	30.9	0	19
1/10/2010	16	29.2	0	23	30.9	0	19
2/10/2010	16	29.2	0	23	30.9	0	19
3/10/2010	16	29.1	1	24	30.8	1	20
4/10/2010	16	29.1	0	24	30.8	0	20
5/10/2010	16	29.1	0	24	30.8	0	20
6/10/2010	16	29.0	1	25	30.7	1	21
7/10/2010	16	28.9	1	26	30.6	1	22
8/10/2010	16	28.7	2	28	30.4	2	24
9/10/2010	7	28.6	1	29	30.3	1	25
10/10/2010	7	28.6	0	29	30.3	0	25
11/10/2010	7	28.6	0	29	30.2	1	26
12/10/2010	7	28.5	1	30	30.2	0	26
13/10/2010	7	28.5	0	30	30.1	1	27
14/10/2010	3	28.4	1	31	30.1	0	27
15/10/2010	3	28.4	0	31	30.1	0	27
16/10/2010	3	28.4	0	31	30.1	0	27
17/10/2010	3	28.3	1	32	30.0	1	28
		GOAF					

3LE-S7-CON

DATE	GED	STATION- S7			STATION- S7A		
		READING	CON	CUM. CON	READING	CON	CUM. CON
21/9/2010	17	39.3	NS	NS	35.5	NS	NS
22/9/2010	17	39.2	1	0	35.5	0	0
23/9/2010	17	39.2	0	0	35.5	0	0
24/9/2010	17	39.2	0	0	35.5	0	0
25/9/2010	17	39.1	1	1	35.4	1	1
26/9/2010	17	39.0	1	2	35.3	1	2
27/9/2010	13	38.8	2	4	35.2	1	3
28/9/2010	13	38.8	0	4	35.2	0	3
29/9/2010	13	38.7	1	5	35.1	1	4
30/9/2010	13	38.5	2	7	35.0	1	5
1/10/2010	13	38.4	1	8	34.9	1	6
2/10/2010	9	38.3	1	9	34.8	1	7
3/10/2010	6	38.2	1	10	34.7	1	8
4/10/2010	6	38.1	1	11	34.6	1	9
5/10/2010	5	37.9	2	13	34.4	2	11
6/10/2010	5	37.9	0	13	34.4	0	11
7/10/2010	5	37.8	1	14	34.3	1	12
8/10/2010	5	37.7	1	15	34.2	1	13
		GOAF					

3LE-S8-CON

DATE	GED	STATION- S8			STATION- S8A		
		READING	CON	CUM. CON	READING	CON	CUM. CON
17/9/2010	21	6.6	NS	NS	3.8	NS	NS
18/9/2010	21	6.6	1	0	3.8	0	0
19/9/2010	18	6.5	1	1	3.7	1	1

20/9/2010	18	6.4	1	2	3.6	1	2
21/9/2010	17	6.4	0	2	3.5	1	3
22/9/2010	17	6.4	0	2	3.5	0	3
23/9/2010	17	6.3	1	3	3.2	3	6
24/9/2010	17	6.3	0	3	3.1	1	7
25/9/2010	17	6.3	0	3	2.9	2	9
26/9/2010	14	6.2	1	4	2.9	0	9
27/9/2010	14	6.0	2	6	2.9	0	9
28/9/2010	11	6.0	0	6	2.9	0	9
29/9/2010	11	5.8	2	8	2.7	2	11
30/9/2010	11	5.6	2	10	2.7	0	11
1/10/2010	11	5.6	0	10	2.5	2	13
2/10/2010	8	5.4	2	12	2.3	2	15
3/10/2010	8	5.3	1	13	2.2	1	16
4/10/2010	8	5.3	0	13	2.2	0	16
5/10/2010	6	5.2	1	14	2.1	1	17
6/10/2010	6	5.1	1	15	2.0	1	18
7/10/2010	6	5.1	0	15	1.8	2	20
8/10/2010	4	5.0	1	16	1.8	0	20
9/10/2010	4	4.8	2	18	1.7	1	21
		GOAF					

3LE-S9-CON

DATE	GED	STATION-S9			STATION- S9A		
		READING	CON	CUM. CON	READING	CON	CUM. CON
24/9/2010	25	14.2	NS	NS	9.0	NS	NS
25/9/2010	25	14.1	1	0	8.9	1	0
26/9/2010	22	14.0	1	1	8.9	0	0
27/9/2010	22	13.9	1	2	8.7	2	2
28/9/2010	22	13.9	0	2	8.7	0	2

29/9/2010	22	13.9	0	2	8.7	0	2
30/9/2010	19	13.7	2	4	8.4	3	5
1/10/2010	19	13.5	2	6	8.3	1	6
2/10/2010	19	13.4	1	7	8.3	0	6
3/10/2010	17	13.4	0	7	8.3	0	6
4/10/2010	17	13.2	2	9	8.0	3	9
5/10/2010	17	13.2	0	9	8.0	0	9
6/10/2010	15	13.0	2	11	7.8	2	11
7/10/2010	12	13.0	0	11	7.7	1	12
8/10/2010	12	12.7	3	14	7.4	3	15
9/10/2010	12	12.5	2	16	7.2	2	17
10/10/2010	10	12.2	3	19	7.0	2	19
11/10/2010	9	12.0	2	21	6.8	2	21
12/10/2010	9	11.7	3	24	6.6	2	23
13/10/2010	5	11.5	2	26	6.6	0	23
14/10/2010	5	11.3	2	28	6.4	2	25
15/10/2010	5	11.1	2	30	6.3	1	26
16/10/2010	5	11.0	1	31	6.1	2	28
17/10/2010	5	10.8	2	33	5.9	2	30
18/10/2010	3	10.8	0	33	5.9	0	30
19/10/2010	3	10.6	2	35	5.8	1	31
20/10/2010	3	10.5	1	36	5.6	2	33
21/10/2010	3	10.3	2	38	5.4	2	35
		GOAF					

3LE-S10-CON

DATE	GED	STATION- S10			STATION- S10A		
		READING	CON	CUM. CON	READING	CON	CUM. CON
2/10/2010	20	15.2	NS	NS	9.0	NS	NS

3/10/2010	18	15.1	1	0	8.9	1	0
4/10/2010	17	15.1	0	0	8.8	1	1
5/10/2010	16	15.1	0	0	8.7	1	2
6/10/2010	13	15.0	1	1	8.5	2	4
7/10/2010	13	14.9	1	2	8.4	1	5
8/10/2010	13	14.7	2	4	8.2	2	7
9/10/2010	13	14.5	2	6	8.0	2	9
10/10/2010	13	14.5	0	6	8.0	0	9
11/10/2010	11	14.3	2	8	7.9	1	10
12/10/2010	11	14.1	2	10	7.7	2	12
13/10/2010	11	14.0	1	11	7.5	2	14
14/10/2010	9	13.8	2	13	7.3	2	16
15/10/2010	9	13.6	2	15	7.2	1	17
16/10/2010	9	13.6	0	15	7.2	0	17
17/10/2010	7	13.5	1	16	7.0	2	19
18/10/2010	7	13.4	1	17	6.9	1	20
19/10/2010	7	13.2	2	19	6.9	0	20
20/10/2010	7	13.1	1	20	6.8	1	21
21/10/2010	7	12.9	2	22	6.6	2	23
22/10/2010	7	12.7	2	24	6.4	2	25
23/10/2010	5	12.6	1	25	6.3	1	26
24/10/2010	5	12.2	4	29	6.0	3	29
25/10/2010	5	12.0	2	31	5.8	2	31
26/10/2010	5	11.9	1	32	5.7	1	32
27/10/2010	5	11.7	2	34	5.6	1	33

		4LE-S3-CON					
DATE	GED	STATION- S3			STATION- S3A		
		READING	CON	CUM. CON	READING	CON	CUM. CON
1/10/2010	20	11.0	IR	IR	8.3	IR	IR

2/10/2010	20	11.0	0	0	8.3	0	0
3/10/2010	20	11.0	0	0	8.3	0	0
4/10/2010	20	11.0	0	0	8.3	0	0
5/10/2010	20	11.0	0	0	8.2	1	1
6/10/2010	18	11.0	0	0	8.2	0	1
7/10/2010	18	10.9	1	1	8.1	1	2
8/10/2010	18	10.9	0	1	8.1	0	2
9/10/2010	18	10.9	0	1	8.1	0	2
10/10/2010	18	10.9	0	1	8.1	0	2
11/10/2010	18	10.9	0	1	8.0	1	3
12/10/2010	16	10.8	1	2	8.0	0	3
13/10/2010	16	10.8	0	2	8.0	0	3
14/10/2010	16	10.7	1	3	7.9	1	4
15/10/2010	14	10.7	0	3	7.9	0	4
16/10/2010	14	10.7	0	3	7.8	1	5
17/10/2010	12	10.5	2	5	7.8	0	5
18/10/2010	12	10.5	0	5	7.7	1	6
19/10/2010	12	10.4	1	6	7.6	1	7
20/10/2010	12	10.4	0	6	7.6	0	7
21/10/2010	12	10.4	0	6	7.6	0	7
22/10/2010	10	10.3	1	7	7.6	0	7
23/10/2010	10	10.3	0	7	7.5	1	8
24/10/2010	8	10.3	0	7	7.5	0	8
25/10/2010	6	10.2	1	8	7.4	1	9
26/10/2010	4	10.1	1	9	7.3	1	10
		GOAF					

4LE-S6-CON

DATE	GED	STATION- S6			STATION- S6A		
		READING	CON	CUM. CON	READING	CON	CUM. CON
3/10/2010	21	25.4	IR	IR	23.4	IR	IR
4/10/2010	21	25.3	1	1	23.4	0	0
5/10/2010	21	25.3	0	1	23.4	0	0
6/10/2010	21	25.3	0	1	23.3	1	1
7/10/2010	21	25.2	1	2	23.3	0	1
8/10/2010	17	25.2	0	2	23.3	0	1
9/10/2010	17	25.0	2	4	23.1	2	3
10/10/2010	17	24.9	1	5	22.9	2	5
11/10/2010	17	24.8	1	6	23.3	1	6
12/10/2010	17	24.8	0	6	23.3	0	6
13/10/2010	17	24.8	0	6	23.3	0	6
14/10/2010	17	24.8	0	6	23.3	0	6
15/10/2010	17	24.7	1	7	23.2	1	7
16/10/2010	14	24.5	2	9	23.1	1	8
17/10/2010	15	24.5	0	9	23.0	1	9
18/10/2010	15	24.5	0	9	23.0	0	9
19/10/2010	15	24.5	0	9	22.9	1	10
20/10/2010	15	24.3	2	11	22.8	1	11
21/10/2010	13	24.1	2	13	22.6	2	13
22/10/2010	13	23.9	2	15	22.5	1	14
23/10/2010	13	23.9	0	15	22.5	0	14
24/10/2010	13	23.9	0	15	22.5	0	14

GOAF

4LE-S10-CON

DATE	GED	STATION- 310			STATION- S10A		
		READING	CON	CUM. CON	READING	CON	CUM. CON
10/12/2010	23	22.2	NS	NS	30.2	NS	NS
11/12/2010	23	22.2	0	0	30.2	0	0
12/12/2010	23	22.2	0	0	30.2	0	0
13/12/2010	17	22.2	0	0	30.0	2	2
14/12/2010	17	22.0	2	2	29.9	1	3
15/12/2010	17	22.0	0	2	29.9	0	3
16/12/2010	17	21.9	1	3	29.8	1	4
17/12/2010	13	21.9	0	3	29.8	0	4
18/12/2010	13	21.8	1	4	29.7	1	5
19/12/2010	13	21.8	0	4	29.5	2	7
20/12/2010	13	21.7	1	5	29.4	1	8
21/12/2010	11	21.5	2	7	29.4	0	8

LEVEL CLOSED

5LE-S2-CON

DATE	GED	STATION- S2			STATION- S2A		
		READING	CON	CUM. CON	READING	CON	CUM. CON
4/10/2010	20	12.6	0	0	23.3	NS	NS
5/10/2010	18	12.6	0	0	23.3	0	0
6/10/2010	18	12.6	0	0	23.3	0	0
7/10/2010	18	12.6	0	0	23.3	0	0
8/10/2010	18	12.6	0	0	23.1	2	2
9/10/2010	18	12.4	2	2	23.1	0	2
10/10/2010	17	12.3	1	3	23.1	0	2
11/10/2010	16	12.3	0	3	23.1	0	2
12/10/2010	14	12.1	2	5	23.0	1	3

13/10/2010	14	12.0	1	6	22.9	1	4
14/10/2010	14	11.9	1	7	22.8	1	5
15/10/2010	14	11.8	1	8	22.8	0	5
16/10/2010	14	11.8	0	8	22.8	0	5
17/10/2010	14	11.7	1	9	22.7	1	6
18/10/2010	11	11.7	0	9	22.7	0	6
19/10/2010	11	11.6	1	10	22.6	1	7
20/10/2010	11	11.5	1	11	22.5	1	8
21/10/2010	11	11.4	1	12	22.4	1	9
22/10/2010	11	11.4	0	12	22.3	1	10
23/10/2010	11	11.4	0	12	22.3	0	10
24/10/2010	9	11.4	0	12	22.2	1	11
25/10/2010	9	11.2	2	14	22.7	PD	PD
26/10/2010	9	11.1	1	15	24.5	NS	NS
27/10/2010	9	10.9	2	17	24.4	1	1
28/10/2010	7	10.8	1	18	24.3	1	2
29/10/2010	7	10.7	1	19	24.3	0	2
30/10/2010	7	10.6	1	20	24.2	1	3
31/10/2010	7	10.6	0	20	24.1	1	4
1/11/2010	7	10.5	1	21	23.8	3	7
2/11/2010	4	10.2	3	24	23.6	2	9
3/11/2010	4	10.0	2	26	23.5	1	10
4/11/2010	4	9.9	1	27	23.4	1	11
5/11/2010	4	9.8	1	28	22.3	1	12
6/11/2010	4	9.7	1	29	22.3	0	12
		GOAF					

6LE-S1-CON

DATE	GED	STATION- S1			STATION- S1A		
		READING	CON	CUM. CON	READING	CON	CUM. CON
27/2/2011	10	27.0	IR	IR	23.0	IR	IR
28/2/2011	10	27.0	0	0	23.0	0	0
1/3/2011	10	27.0	0	0	23.0	0	0
2/3/2011	10	26.9	1	1	22.9	1	1
3/3/2011	10	26.9	0	1	22.9	0	1
4/3/2011	10	26.8	1	2	22.8	1	2
5/3/2011	10	26.7	1	3	22.8	0	2
6/3/2011	10	26.7	0	3	22.7	1	3
7/3/2011	10	26.5	2	5	22.5	2	5
8/3/2011	10	26.5	0	5	22.5	0	5
9/3/2011	6	26.3	2	7	22.5	0	5
10/3/2011	6	26.2	1	8	22.5	0	5
11/3/2011	6	26.2	0	8	22.5	0	5
12/3/2011	4	26.1	1	9	22.4	1	6
13/3/2011	4	26.1	0	9	22.3	1	7
14/3/2011	4	26.0	1	10	22.3	0	7
15/3/2011	4	26.0	0	10	22.1	2	9

GOAF

7LE-S10-CON

DATE	GED	STATION- S10			STATION- S10A		
		READING	CON	CUM. CON	READING	CON	CUM. CON
20/6/2011	20	8.5	NS	NS	3.8	NS	NS
21/6/2011	20	8.5	0	0	3.8	0	0
22/6/2011	18	8.5	0	0	3.8	0	0

23/6/2011	18	8.4	1	1	3.7	1	1
24/6/2011	18	8.3	1	2	3.6	1	2
25/6/2011	16	8.3	0	2	3.5	1	3
26/6/2011	16	8.3	0	2	3.5	0	3
27/6/2011	14	8.1	2	4	3.4	1	4
28/6/2011	14	8.0	1	5	3.2	2	6
29/6/2011	12	8.0	0	5	3.1	1	7
30/6/2011	10	7.9	1	6	3.1	0	7
1/7/2011	10	7.8	1	7	3.1	0	7
2/7/2011	10	7.8	0	7	3.0	1	8
3/7/2011	8	7.6	2	9	2.9	1	9
4/7/2011	8	7.4	2	11	2.7	2	11
5/7/2011	6	7.3	1	12	2.7	0	11
6/7/2011	6	7.2	1	13	2.7	0	11
7/7/2011	6	7.1	1	14	2.6	1	12
8/7/2011	4	7.1	0	14	2.6	0	12
9/7/2011	4	7.0	1	15	2.5	1	13
10/7/2011	4	7.0	0	15	2.5	0	13
11/7/2011	2	7.0	0	15	2.5	0	13
		GOAF					

APPENDIX- II: LOAD ON ROOF BOLTS SUPPORT

1LE LOAD CELL 1

DATE	GED m	GREEN LOAD	YELLOW LOAD	RED LOAD	TOTAL LOAD	AVG LOAD	DAILY VAR. LOAD	CUM LOAD	REMARKS
4/7/2010	10	22.89	7.37	-22.56	7.700	2.567	SETTING	SETTING	
5/7/2010	10	24.12	-7.74	-8.27	8.110	2.703	0.136	0.136	
6/7/2010	10	22.33	-8.16	-6.12	8.055	2.685	-0.018	0.118	
7/7/2010	10	23.15	-7.90	-7.37	7.880	2.627	-0.062	0.056	
10/7/2010	8	23.87	-5.64	-10.10	8.130	2.710	0.087	0.143	
11/7/2010	6	25.45	-7.79	-9.58	8.080	2.693	-0.017	0.126	
12/7/2010	6	22.12	-7.39	-6.77	7.960	2.653	-0.040	0.086	
13/7/2010	6	20.98	-9.11	-4.40	7.470	2.490	-0.163	-0.077	
14/7/2010	4	21.14	-9.11	-4.61	7.420	2.473	-0.016	-0.093	
15/7/2010	3	24.48	-7.25	-10.07	7.160	2.387	-0.086	-0.179	
16/7/2010	2	21.88	-7.31	-7.29	7.280	2.427	0.040	-0.139	
17/7/2010	2	25.12	-6.69	-11.42	7.013	2.338	-0.088	-0.227	
			GOAF						

2LE-LOAD CELL 3

DATE	GED m	GREEN LOAD	YELLOW LOAD	RED LOAD	TOTAL LOAD	AVG LOAD	DAILY VAR. LOAD	CUM LOAD	REMARKS
30/8/2010	18	0.70	5.20	17.40	23.300	7.767	SETTING	SETTING	
5/31/2011	18	0.68	5.18	17.30	23.160	7.720	-0.047	-0.047	
6/1/2011	18	0.66	5.28	17.39	23.330	7.777	0.057	0.010	
6/2/2011	18	0.65	5.30	17.26	23.210	7.737	-0.040	-0.030	
6/3/2011	15	0.64	5.35	17.30	23.290	7.763	0.027	-0.003	

6/4/2011	15	0.69	5.45	17.38	23.520	7.840	0.077	0.073	
6/5/2011	15	0.70	5.55	17.45	23.700	7.900	0.060	0.133	
6/6/2011	15	0.73	5.68	17.49	23.900	7.967	0.067	0.200	
6/7/2011	13	0.79	5.69	17.51	23.990	7.997	0.030	0.230	
6/9/2011	13	0.80	5.71	17.70	24.210	8.070	0.073	0.303	
6/10/2011	13	0.85	5.75	17.75	24.350	8.117	0.047	0.350	
6/11/2011	11	0.85	5.71	17.75	24.310	8.103	-0.013	0.337	
6/12/2011	11	0.87	5.76	17.75	24.380	8.127	0.023	0.360	
6/13/2011	10	1.48	2.70	1.03	5.210	1.737	RESETTING	RESETTING	
6/14/2011	10	7.53	2.83	0.74	11.100	3.700	1.963	1.963	
6/15/2011	8	8.95	3.16	1.38	13.490	4.497	0.797	2.760	
6/16/2011	8	12.32	5.22	2.20	19.740	6.580	2.083	4.843	
6/17/2011	8	11.50	4.30	1.70	17.500	5.833	-0.747	4.097	
6/18/2011	8	11.40	4.16	1.58	17.140	5.713	-0.120	3.977	
6/19/2011	6	8.05	3.60	1.23	12.880	4.293	-1.420	2.557	
6/20/2011	6	10.64	3.54	1.20	15.380	5.127	0.833	3.390	
6/21/2011	6	10.72	3.34	1.48	15.540	5.180	0.053	3.443	
6/22/2011	6	10.72	3.36	1.50	15.580	5.193	0.013	3.457	
6/23/2011	4	10.71	3.36	1.57	15.640	5.213	0.020	3.477	
6/24/2011	4	10.72	3.37	2.00	16.090	5.363	0.150	3.627	
6/26/2011	4	10.72	3.40	2.08	16.200	5.400	0.037	3.663	

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3LE- LOAD CELL 5

DATE	GED m	GREEN LOAD	YELLOW LOAD	RED LOAD	TOTAL LOAD	AVG LOAD	DAILY VAR. LOAD	CUM LOAD	REMARKS
24/10/2010	6	8.21	1.11	12.35	21.67	7.223	SETTING	SETTING	

25/10/2010	6	8.55	1.03	12.14	21.72	7.240	0.017	0.017	
26/10/2010	6	8.32	1.66	12.25	22.23	7.410	0.170	0.187	
27/10/2010	4	24.59	-3.10	23.62	45.11	15.037	RESETTING	RESETTING	
28/10/2010	4	8.34	-3.31	6.28	11.31	3.770	RESETTING	RESETTING	
29/10/2010	3	12.72	-4.28	19.54	27.98	9.327	RESETTING	RESETTING	
30/10/2010	3	7.06	-3.91	15.88	19.03	6.343	RESETTING	RESETTING	
31/10/2010	2	7.74	-3.76	15.23	19.21	6.403	0.060	0.247	
1/11/2010	2	7.41	-3.25	15.06	19.22	6.407	0.003	0.250	
2/11/2010	2	7.89	-3.64	15.36	19.61	6.537	0.130	0.380	
			GOAF						

DATE	GED m	GREEN LOAD	YELLOW LOAD	RED LOAD	TOTAL LOAD	AVG LOAD	DAILY VAR. LOAD	CUM LOAD	REMARKS
29/11/2010	27	13.59	-2.64	4.32	15.270	5.090	SETTING	SETTING	
30/11/2010	26	18.49	-5.12	5.30	18.670	6.223	1.133	1.133	
1/12/2010	26	18.87	-5.78	5.95	19.040	6.347	0.123	1.257	
2/12/2010	26	19.24	-6.32	6.53	19.450	6.483	0.137	1.393	
3/12/2010	24	19.78	-7.11	6.79	19.460	6.487	0.003	1.397	
4/12/2010	24	20.24	-8.44	7.35	19.150	6.383	-0.103	1.293	
5/12/2010	22	20.95	-9.67	8.11	19.390	6.463	0.080	1.373	
6/12/2010	22	21.38	-10.43	8.26	19.210	6.403	-0.060	1.313	
7/12/2010	22	22.35	-11.37	8.38	19.360	6.453	0.050	1.363	
8/12/2010	20	23.46	-12.54	8.64	19.560	6.520	0.067	1.430	
9/12/2010	20	24.75	-13.61	8.91	20.050	6.683	0.163	1.593	
10/12/2010	19	25.48	-14.57	9.14	20.052	6.684	0.001	1.594	
11/12/2010	19	26.45	-15.34	9.52	20.630	6.877	0.193	1.787	
12/12/2010	19	27.42	-16.82	9.86	20.458	6.819	-0.057	1.729	
13/12/2010	19	28.39	-17.63	10.17	20.926	6.975	0.156	1.885	

14/12/2010	19	29.35	-18.38	10.35	21.324	7.108	0.133	2.018	
15/12/2010	19	29.84	-18.04	10.42	22.220	7.407	0.299	2.317	
16/12/2010	19	30.11	-18.20	10.51	22.420	7.473	0.067	2.383	
17/12/2010	19	30.36	-19.02	10.83	22.170	7.390	-0.083	2.300	
18/12/2010	19	30.45	-19.24	11.04	22.250	7.417	0.027	2.327	
19/12/2010	19	30.58	-19.47	11.25	22.360	7.453	0.037	2.363	
20/12/2010	19	31.02	-19.17	11.42	23.270	7.757	0.303	2.667	
21/12/2010	19	31.14	-19.28	11.38	23.240	7.747	-0.010	2.657	
22/12/2010	19	31.26	-19.17	11.36	23.450	7.817	0.070	2.727	
23/12/2010	19	31.54	-19.17	11.52	23.898	7.966	0.149	2.876	
24/12/2010	19	31.57	-19.13	11.60	24.034	8.011	0.045	2.921	
25/12/2010	19	31.98	-19.10	11.68	24.554	8.185	0.173	3.095	
26/12/2010	16	32.09	-19.06	11.75	24.782	8.261	0.076	3.171	
27/12/2010	16	32.42	-19.04	11.83	25.210	8.403	0.143	3.313	
28/12/2010	16	32.51	-19.00	11.91	25.414	8.471	0.068	3.381	
29/12/2010	16	32.49	-18.97	11.98	25.504	8.501	0.030	3.411	
30/12/2010	16	33.57	-18.94	12.06	26.694	8.898	0.397	3.808	
31/12/2010	16	33.62	-18.90	12.14	26.854	8.951	0.053	3.861	
1/1/2011	16	33.75	-18.87	12.21	27.094	9.031	0.080	3.941	
2/1/2011	16	33.84	-18.84	12.29	27.294	9.098	0.067	4.008	
3/1/2011	16	33.98	-18.80	12.37	27.544	9.181	0.083	4.091	
4/1/2011	16	34.06	-18.77	12.45	27.734	9.245	0.063	4.155	
5/1/2011	16	34.38	-18.74	12.52	28.162	9.387	0.143	4.297	
6/1/2011	16	34.60	-18.71	12.49	28.381	9.460	0.073	4.370	
7/1/2011	16	34.81	-18.67	12.68	28.818	9.606	0.146	4.516	
8/1/2011	16	35.03	-18.64	12.77	29.163	9.721	0.115	4.631	
9/1/2011	16	35.25	-18.61	12.83	29.474	9.825	0.104	4.735	
10/1/2011	16	35.47	-18.57	12.92	29.815	9.938	0.114	4.848	
11/1/2011	16	35.69	-18.61	12.98	30.060	10.020	0.082	4.930	
12/1/2011	16	35.90	-18.51	13.06	30.458	10.153	0.133	5.063	

13/1/2011	16	36.06	-18.47	13.14	30.724	10.241	0.089	5.151	
14/1/2011	16	36.11	-18.44	13.22	30.884	10.295	0.053	5.205	
15/1/2011	16	36.56	-18.41	13.29	31.442	10.481	0.186	5.391	
16/1/2011	16	36.48	-18.38	13.37	31.474	10.491	0.011	5.401	
17/1/2011	16	36.61	-18.34	13.45	31.714	10.571	0.080	5.481	
18/1/2011	16	36.78	-18.31	13.52	31.994	10.665	0.093	5.575	
19/1/2011	16	36.92	-18.28	13.60	32.244	10.748	0.083	5.658	
20/1/2011	16	37.05	-18.24	13.68	32.484	10.828	0.080	5.738	
21/1/2011	16	37.14	-18.21	13.75	32.684	10.895	0.067	5.805	
22/1/2011	16	37.25	-18.18	13.83	32.904	10.968	0.073	5.878	
23/1/2011	7	37.42	-18.14	13.91	33.184	11.061	0.093	5.971	
24/1/2011	7	38.07	-18.11	13.99	33.944	11.315	0.253	6.225	
25/1/2011	7	38.74	-18.08	14.06	34.722	11.574	0.259	6.484	
26/1/2011	7	38.96	-18.05	14.14	35.050	11.683	0.109	6.593	
27/1/2011	7	38.93	-18.01	14.22	35.134	11.711	0.028	6.621	
28/1/2011	3	39.39	-17.98	14.29	35.706	11.902	0.191	6.812	
29/1/2011	3	39.64	-17.95	14.37	36.064	12.021	0.119	6.931	
30/1/2011	3	39.83	-17.91	14.45	36.362	12.121	0.099	7.031	
31/1/2011	3	40.16	-17.88	14.52	36.804	12.268	0.147	7.178	

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5LE- LOAD CELL- 7

DATE	GED m	GREEN LOAD	YELLOW LOAD	RED LOAD	TOTAL LOAD	AVG LOAD	DAILY VAR. LOAD	CUM LOAD	REMARKS
25/11/2010	14	4.09	-1.52	6.75	9.320	3.107	SETTING	SETTING	
26/Aug/11	14	7.73	-0.99	8.03	14.770	4.923	1.817	1.817	
27/Aug/11	14	8.14	-0.92	8.62	15.840	5.280	0.357	2.173	
28/Aug/11	14	8.32	-0.99	8.75	16.080	5.360	0.080	2.253	

29/Aug/11	14	8.91	-0.72	9.71	17.900	5.967	0.607	2.860	
30/Aug/11	14	9.16	-0.72	11.18	19.620	6.540	0.573	3.433	
31/Aug/11	14	9.98	-1.74	11.54	19.780	6.593	0.053	3.487	
1/Sep/11	14	10.37	-1.46	11.32	20.230	6.743	0.150	3.637	
3/Sep/11	12	11.03	-1.41	13.87	23.490	7.830	1.087	4.723	
4/Sep/11	10	12.23	-0.23	14.21	26.210	8.737	0.907	5.630	
5/Sep/11	10	12.40	-1.52	15.52	26.400	8.800	0.063	5.693	
6/Sep/11	10	12.61	-1.46	15.93	27.080	9.027	0.227	5.920	
7/Sep/11	10	12.70	-1.36	16.21	27.550	9.183	0.157	6.077	
8/Sep/11	10	13.04	-1.36	17.14	28.820	9.607	0.423	6.500	
9/Sep/11	10	13.17	-1.53	17.62	29.263	9.754	0.148	6.648	
10/Sep/11	8	13.54	-1.62	18.19	30.108	10.036	0.282	6.929	
11/Sep/11	8	13.56	-1.71	18.75	30.600	10.200	0.164	7.093	
12/Sep/11	8	13.75	-1.15	19.32	31.914	10.638	0.438	7.531	
13/Sep/11	6	13.94	-1.88	19.88	31.938	10.646	0.008	7.539	
14/Sep/11	6	14.13	-1.97	20.56	32.721	10.907	0.261	7.800	
15/Sep/11	6	14.32	-2.06	21.01	33.275	11.092	0.185	7.985	
16/Sep/11	6	14.52	-2.56	21.57	33.531	11.177	0.085	8.070	
17/Sep/11	6	14.71	-2.24	22.14	34.612	11.537	0.360	8.431	
18/Sep/11	6	14.90	-2.32	22.65	35.227	11.742	0.205	8.636	
19/Sep/11	6	15.09	-2.41	23.27	35.949	11.983	0.241	8.876	
20/Sep/11	6	15.28	-2.76	23.83	36.357	12.119	0.136	9.012	
21/Sep/11	6	15.48	-2.59	24.40	37.286	12.429	0.310	9.322	
22/Sep/11	4	15.67	-2.68	24.70	37.693	12.564	0.135	9.458	
23/Sep/11	4	15.86	-2.76	25.53	38.623	12.874	0.310	9.768	
			GOAF						

6LE LOAD CELL- 10

DATE	GED m	GREEN LOAD	YELLOW LOAD	RED LOAD	TOTAL LOAD	AVG LOAD	DAILY VAR. LOAD	CUM LOAD	REMARKS
28/2/2011	21	10.79	-2.17	3.10	11.720	3.907	SETTING	SETTING	
11/29/2011	21	9.80	-1.10	2.81	11.510	3.837	-0.070	-0.070	
11/30/2011	19	9.42	-1.72	3.52	11.220	3.740	-0.097	-0.167	
12/1/2011	19	9.11	-2.54	3.17	9.740	3.247	-0.493	-0.660	
12/2/2011	19	9.78	-2.11	2.93	10.600	3.533	0.287	-0.373	
12/3/2011	16	10.12	-1.76	2.54	10.900	3.633	0.100	-0.273	
12/4/2011	16	10.54	-1.91	2.72	11.350	3.783	0.150	-0.123	
12/5/2011	14	10.82	-2.66	3.11	11.270	3.757	-0.027	-0.150	
12/6/2011	14	10.16	-2.01	3.63	11.780	3.927	0.170	0.020	
12/7/2011	14	9.79	-1.33	3.38	11.840	3.947	0.020	0.040	
12/8/2011	12	9.92	-1.51	3.91	12.320	4.107	0.160	0.200	
12/9/2011	12	10.12	-1.18	3.72	12.660	4.220	0.113	0.313	
12/10/2011	10	10.18	-0.83	3.18	12.530	4.177	-0.043	0.270	
12/11/2011	10	10.63	-0.12	3.24	13.750	4.583	0.407	0.677	
12/12/2011	8	10.85	1.56	1.88	14.290	4.763	0.180	0.857	
12/13/2011	6	10.27	1.72	1.96	13.950	4.650	-0.113	0.743	
12/14/2011	6	10.39	1.14	1.28	12.810	4.270	-0.380	0.363	
12/15/2011	4	9.71	1.37	1.11	12.190	4.063	-0.207	0.157	
12/16/2011	4	9.56	-0.82	3.07	11.810	3.937	-0.127	0.030	

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7LE LOAD CELL- 1

DATE	GED m	GREEN LOAD	YELLOW LOAD	RED LOAD	TOTAL LOAD	AVG LOAD	DAILY VAR. LOAD	CUM LOAD	REMARKS
27/10/2010	10	3.35	1.15	2.43	6.930	2.310	SETTING	SETTING	
7/28/2011	10	3.35	3.78	2.38	9.510	3.170	RESETTING	RESETTING	
7/29/2011	10	3.22	3.96	2.38	9.560	3.187	0.017	0.017	
7/30/2011	10	3.22	4.00	2.29	9.510	3.170	-0.017	0.000	
7/31/2011	8	3.17	3.96	2.29	9.420	3.140	-0.030	-0.030	
8/1/2011	8	3.17	3.91	2.29	9.370	3.123	-0.017	-0.047	
8/2/2011	6	3.22	4.00	2.34	9.560	3.187	0.063	0.017	
8/3/2011	6	3.31	4.05	2.38	9.740	3.247	0.060	0.077	
8/4/2011	6	3.31	4.00	2.38	9.690	3.230	-0.017	0.060	
8/5/2011	6	3.26	4.00	2.38	9.640	3.213	-0.017	0.043	
8/6/2011	4	3.44	3.96	2.48	9.880	3.293	0.080	0.123	
8/7/2011	4	3.40	3.90	2.43	9.730	3.243	-0.050	0.073	

GOAF

APPENDIX-III: ROCK BOLT PULL TEST DATA

RESULTS OF PULL TESTS ON ROOFBOLTS

DATE	LOCATION MINE, SECTION	ROCK AT ANCHOR	LOAD (tons)	REMARK
10.11.2010	1LE/14ND	MSST	18	Nut stripped
10.11.2010	1LE/15ND	MSST	17.5	Holding
10.11.2010	1LE/16ND	MSST	14	Holding
10.11.2010	1LE/17ND	MSST	15	Holding
10.11.2010	1LE/18ND	FGSST	16	Holding
10.11.2010	2LE/13ND	MGSST	14	Nut stripped
10.11.2010	2LE/14ND	MGSST	14	Holding
10.11.2010	2LE/15ND	MGSST	15	Holding
10.11.2010	2LE/16ND	MSST	16	Holding
10.11.2010	2LE/17ND	MSST	18	Holding
10.11.2010	3LE/12ND	MSST	17.5	Nut stripped-pinch thread bolt
10.11.2010	2LE/13ND	MSST	15.5	Holding
14.01.2011	3LE/12ND	MSST	15	Holding
14.01.2011	3LE/13ND	MSST	16	Holding
14.01.2011	3LE/15ND	MSST	17.5	Nut stripped
06.04.2011	4LE/11ND	FGSST	15	Holding-torque-18 MPT
06.04.2011	4LE/13ND	FGSST	14	Holding
06.04.2011	4LE/15ND	MGSST	16	Holding
06.04.2011	4LE/17ND	MGSST	15.5	Holding
14.06.2011	5LE/12ND	MGSST	12	Holding -pinch thread bolt
14.06.2011	5LE/15ND	MGSST	9	Resin rock failure-250 mm resin encapsulation

14.06.2011	5LE/17ND	MSST	11	Holding -pinch thread bolt
14.06.2011	6LE/12ND	MSST	14	Holding-torque-22 MPT
14.06.2011	6LE/14ND	MSST	15	Holding
14.06.2011	6LE/16ND	MSST	16	Holding
14.06.2011	6LE/17ND	FGSST	12	Holding -pinch thread bolt

DATE	LOCATION MINE, SECTION	ROCK AT ANCHOR	LOAD (tonne)	REMARK
12.08.2010	2LE/15ND	MSST	12	Holding-torque-22 MPT
12.08.2010	3LE/12ND	MSST	15	Holding
12.08.2010	4LE/13ND	FGSST	11	Holding
22.12.2010	3LE/15ND	FGSST	14	Holding
22.12.2010	5LE/12ND	MSST	15.5	Holding -pinch thread bolt
22.12.2010	6LE/13ND	CGSST	12	Holding
16.02.2011	3LE/14ND	MSST	14	Holding
16.02.2011	4LE/12ND	MGSST	09	Resin rock failure- 250 mm resin encapsulation
16.02.2011	5LE/14ND	MSST	14.5	Holding
20.06.2011	6LE/13ND	FGSST	13	Holding
20.06.2011	6LE/15ND	FGSST	14	Holding
20.06.2011	7LE/10ND	MSST	16.5	Holding -pinch thread bolt
20.06.2011	7LE/16ND	FGSST	14	Holding

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1. Singh S.K., Jayanthu S., and Singh Gopal, Strata Control Problem in S.E.C.L. for Mass Exploitation of Underground Coal Deposits. Proceedings of Geomintech Symposium on New Equipment New Technology-Management and Safety in Mines and Mineral Based Industries,[ENTMS2009],Dtd.11-12 May 2009,pp62-66,Bhubaneswar,Orissa, India.
2. Singh S.K., Jayanthu S., Singh T.N. and Singh D.P., Mine Safety vis-à-vis Geotechnical Investigations, Workshop on Safety-2009, Dtd 5-6 Nov 2009, Kolkata, India.
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